Development of an Integrated Mining and Processing Optimization System

To the Faculty of Geosciences, Geo-Engineering and Mining

of the Technische Universität Bergakademie Freiberg

approved

DISSERTATION

to attain the academic degree of

Doktor-Ingenieur

(Dr.-Ing)

submitted

by M.Sc. Ayman Abdelfattah Mahmoud, Ahmed

born on the 25.09.1969 in Red Sea, Egypt

Reviewers: Prof. Dr. Carsten Drebenstedt (TU-Freiberg)

Prof. Dr.-Ing. Christian Niemann Delius (RWTH Aachen)

Prof. Dr. Eng. Mohamed Elwageeh (Cairo University)

Freiberg, April 2013

Acknowledgement

I would like to express my greatest thanks and gratitude to my supervisor Prof. Dr. Carsten Drebenstedt for his scientific supervision, constructive criticism, and dedicated guidance during my study in Germany. I gratefully acknowledge also his kindness, support, and helpful advices.

Many thanks are addressed to Prof. Dr. Christian Niemann Delius (RWTH Aachen) and Prof. Dr. Mohamed Elwageeh (Cairo University) for accepting of the scientific reviewing of my thesis.

I am also greatly thankful to my country (Egypt) for awarding me the scholarship and continual supporting - despite the previous circumstances during the revolution - to finish my doctoral work in TU Bergakademie Freiberg.

Many thanks and gratitude are also to all my colleagues at the Chair of Surface Mining for their kindness and encouragement during my study. I especially thank my colleague and friend Richard Eichler very much for his help and support during the last two years of my stay in Freiberg. I also do not forget to express my deep thanks to my precedent colleague Martin Kreßner for his kind assistance during the first period of my study in Germany.

I also thank all my German, Arabic and Egyptian friends in Freiberg for their friendship, encouragement, and valuable discussions.

I address my sincere thanks to my mother and all members in my big family in Egypt for their continuous support.

Finally, I thank my wife Usra Ahmed and my three sons Shehab, Belal, and Mostafa for their great understanding, patience, and encouragement during all the years we spent in Germany for studying and finishing my dissertation.

Table of Content

List of Figures

List of Tables

List of symbols and Abbreviations

a	Drag resistance factor
A_f	Truck front area, (m^2)
B	Blasting burden, (m)
b	Rolling and gradient factor
BFC	Brake fuel consumption, (kg/kWh)
BI	Brittleness index
$BM_{\$}$	Ball mill price, (M$)
$C_{b.fix}$	Blasting fixed cost, ($/hole)
C_d	Drilling cost, ($/m)
$C_{d.fix}$	Drilling fixed cost, ($/m)
$C_{d.op}$	Drilling operating cost, ($/h)
$C_{d.var}$	Drilling variable cost, ($/m)
$C_{op.c\&g}$	Crushing & grinding operating cost, (M$)
$C_{op.l\&h}$	Loading & hauling operating cost, (M$)
D	Blast hole diameter, (mm)
DT	Drilling torque, (t.m/h);
$Elec_{\$}$	Electric energy price, ($/kWh)
eq_{CO2}	CO_2-Equivalent, (kg/kWh)
$f_{\$}$	Fuel price, ($/kg)
F_{max}	Drilling-rig maximum thrust, (t)
$fX_{c.g}$	80% passing size feed, coarse grinding, (μm)
$fX_{p.c}$	80% passing size feed, primary crushing, (μm)
g	Acceleration of gravity, (m/s^2)
H	Bench height, (m)
i	Year number
$JC_{\$}$	Jaw crusher price, (M$)

K_{ass}	Assessment factor
$K_{b.pass,c.g}$	Coarse grinding bypass factor
$K_{b.pass,p.c}$	Primary crushers bypass factor
K_{drag}	Drag coefficient
K_{f}	Filling factor
K_{hard}	Hardness factor
K_{joint}	Joint factor
K_{rock}	Rock factor
K_{roll}	Coefficient of rolling resistance
L	Total charge length, (m)
L_{c}	Distance to plant, (m)
L_{d}	Distance to dumping, (m)
$M_{an.ore}$	Ore annual production, (t)
$M_{b.ore}$	Ore produced per blast, (t)
$M_{b.pass,c.g}$	Coarse grinding bypass, (t/h)
$M_{b.pass,f.g}$	Fine grinding bypass, (t/h)
$M_{b.pass,p.c}$	Primary crushers bypass, (t/h)
$M_{bulk,tr}$	Truck bulk transfer rate, (t/h)
m_{c}	Metal content, (%)
$M_{c.c}$	Cyclone cluster feeding rate, (t/h)
$M_{c.g}$	Coarse grinding feeding rate, (t/h)
$M_{c.screen}$	Coarse screen feeding rate, (t/h)
$M_{CO2,tr}$	Truck CO_2-Emission rate, (kg/h)
M_{e}	Truck empty weight, (t)
M_{ex}	Quantity of explosive per hole, (kg)
$M_{f,trip}$	Trip fuel consumption per truck, (kg)
M_{g}	Truck gross weight, (t)
M_{mill}	Total milled ore rate, (t/h)
M_{o}	Blasted tonnage per bore hole, (t)
$M_{p.c}$	Primary crushers feeding rate, (t/h)
M_{pay}	Truck pay load, (t)

m_r	Milling recovery, (%)
M_{res}	Ore reserve, (t)
M_{rr}	Metal recovery rate, (kg/h)
$M_{s.screen}$	Secondary screen feeding rate, (t/h)
$Msp._{CO2,M}$	Specific CO_2 emission (mine), (kg/t)
$Msp._{CO2,P}$	Specific CO_2 emission (plant), (kg/t)
$M_{sp.ex}$	Powder factor, (kg/m^3)
$M_{thr,c.g}$	Coarse grinding throughput, (t/h)
$M_{thr,f.g}$	Fine grinding throughput, (t/h)
$M_{thr,p.c}$	Primary crushers throughput, (t/h)
$M_{tot.CO2}$	Total CO_2 emission amount, (10^3*t)
$M_{tot.rec}$	Total metal recovery amount, (t)
M_{trans}	Ore transfer to plant, (t/h)
N	Revolution number, (rpm)
n	Uniformity exponent
n_{bh}	Blasting holes number
n_{blast}	Blasting frequency
$n_{buckets}$	Shovel loading rate, (1/h)
n_d	Annual working days
n_h	Daily working hours, (h)
n_I	Number of investment times
$n_{J.c,real}$	Number of *Jaw Crushers* in duty
$n_{J.c,theo}$	Theoretical number of *Jaw Crushers*
n_{ore}	Ore-types number
n_{pass}	Number of bucket passes per truck
NPV	Net present value, (M$)
$n_{S.m,real}$	Number of *SAG Mills* in duty
$n_{S.m,theo}$	Theoretical number of *SAG Mills*
n_{sc}	Number of scenarios
n_{sh}	Number of shovels
n_{tr}	Number of trucks

$n_{trip,tr}$	Truck trip frequency, (1/h)
n_y	Mine life, (Years)
$P_{avail,c.g}$	Coarse grinding available power, (kW)
$P_{avail,p.c}$	Primary crushing available power, (kW)
$P_{c.g}$	Power required, coarse grinding, (kW)
P_e	Empty truck power, (kW)
P_l	Loaded truck power, (kW)
$P_{p.c}$	Power required, primary crushing,(kW)
PR	Penetration rate, (m/h)
P_t	Total power required, (kW)
$pX_{c.g}$	80% passing size product, coarse grinding, (μm)
$pX_{p.c}$	80% passing size product, primary crushing, (μm)
r	Discount rate, \$/y
R	Percentage smaller than X
RDI	Rock density index
RMD	Rock mass description
$R\,M_{J.c}$	*Jaw Crusher* rated capacity, (t/h)
$R\,M_{S.m}$	*SAG Mill* rated capacity, (t/h)
$RP_{J.c}$	*Jaw Crusher* rated power, (kW)
$RP_{S.m}$	*SAG Mill* rated power, (kW)
RWS	Relative weight strength, explosive
S	Blast hole spacing, (m)
SC_b	Specific blasting cost, (\$/t)
SC_d	Specific drilling cost, (\$/t)
$SC_{d\&b}$	Specific drilling and blasting cost, (\$/t);
SC_{energy}	Electric energy specific cost, (\$/t)
SC_f	Fuel consumption specific cost, (\$/t)
$SC_{f\&c}$	Floatation and other specific cost, (\$/t)
$SC_{l\&h}$	Loading and hauling specific cost, (\$/t)
SC_M	Total mining specific costs, (\$/t)
$SC_{m\&p}$	Total mining and processing specific cost, (\$/t)

SC_P	Processing specific cost, ($/t)
$SE_{c.g}$	Coarse grinding specific energy, (kWh/t)
SE_d	Drilling specific energy, (t.m/m3)
$SE_{f.g}$	Fine grinding specific energy, (kWh/t)
$SE_{p.c}$	Primary crushing specific energy, (kWh/t)
SE_{Plant}	Plant specific energy, (kWh/t)
$sh_{\$}$	Shovel price, (M$)
$SM_{\$}$	SAG mill price, (M$)
SM_{fuel}	Specific fuel consumption, (kg/t)
S_r	Stripping ratio, (t/t), Dmnl
t_{cyc}	Total cycle time, (min)
t_d	Dumping time, (min)
t_l	Loading time, (min)
t_{man}	Maneuver time, (min)
$tr_{\$}$	Truck price, (M$)
t_{te}	Travelling time (empty), (min)
t_{tl}	Travelling time (loaded), (min)
t_{tr}	Travelling time, (min)
t_w	Waiting time, (min)
$U_{c.g}$	Coarse grinding facility utilization, (%)
U_{fleet}	Fleet utilization, (%)
$U_{p.c}$	Primary crushing facility utilization, (%)
V_{bucket}	Bucket capacity, (m^3)
v_e	Empty travelling velocity, (m/min)
V_{excav}	Required excavation rate , (m^3/h);
v_l	Loaded travelling velocity, (m/min)
V_o	Bank volume, (m^3)
V_{sh}	Volume excavated rate , (m^3/h)
V_{tr}	Truck capacity, (m^3)
w	Drilling accuracy, (m)
$WI_{c.g}$	Coarse grinding work index, (kWh/t)

$WI_{p.c}$	Primary crushing work index, (kWh/t)
X	Rock size, (cm)
X_{50}	Mean fragmentation size, (cm)
X_l	Average liberation size, (μm)
$\$_{con}$	Concentrated ore price, (\$/t)
$\$_{cost}$	Annual costs, (M\$)
$\$_{ex}$	Explosive price, (\$/kg)
$\$_f$	Total annual cash flow, (M\$)
$\$_{f,i}$	Discrete annual cash flow, (M\$)
$\$_I$	Investments and expenses before production, (M\$)
$\$I_{c\&g}$	Crushing & grinding capital investments, (M\$)
$\$I_{l\&h}$	Loading & hauling capital investments, (M\$)
$\$_{in}$	Annual income, (M\$)
Π	Liberation probability
γ_{fleet}	Fleet availability, (%)
Δ	Sub-drilling, (m)
δ_x	Liberation size standard deviation, (μm)
ε	Discontinuity (spacing), (m)
η_{fleet}	Fleet efficiency
η_l	Loading efficiency
θ	Road gradient, (rad)
ρ	Bank rock density, (t/m^3)
ρ_{air}	Air density, (kg/m^3)
ρ_b	Muck-pile bulk density, (t/m^3)
σ_C	Uniaxial compressive strength, (MPa)
ς_j	Ore-type sharing in the ore deposit, (%)
σ_T	Tensile strength, (MPa)
ω	Swilling factor

List of Appendices

1. Justification and Importance of the Mine Planning Optimization

1.1 Introduction

The history of mining is traced back to the ancient Egyptians, who operated malachite mines at *Wady Maghareh* on the Sinai Peninsula. Today millions of people are employed in the mining industry throughout the world. For example, in the USA alone, in 2008, around 675,000 people are employed in the natural resources and mining sector [15, 84].

Each year, billions of dollars are spent to produce various types of equipment and technology for use by the mining industry throughout the world; and this expenditure is increasing rapidly. For example, in 2004 American mining equipment companies shipped around $1.4 billion worth of goods; and a year later, in 2005, the figure jumped to $2 billion and $4.7 billion in 2009, [31, 34, 134].

Nowadays, the competitive global economy is forcing mining companies around the world to optimize its operations through increased mechanization and automation and good general planning.

Good mine planning for a new mine project, for example, may involve:

1) *Before-production planning* as: a) feasibility study for research, prospecting, exploration and all pre-starting expenses; and b) the preparation stage as mining method planning, machinery investments and proprietary assets expenses.

2) *During-production planning* as mining, processing and marketing planning and costing.

3) *Post-production planning* as the mine closure and reclamation planning and costing after the mining ceases.

1.2 Urgent need for general mine planning optimization

1.2.1 Overall costly low-grade ore deposits

In most countries the easily found deposits, which are cropping out at the surface, had nearly all been found and exhausted. The potential for finding new resources of high quality deposits that readily accessible, high grade, big tonnage ore bodies and preferably in a politically stable country is very small. Figure 1 shows the copper industry ore grade during 1770- 2007 [27].

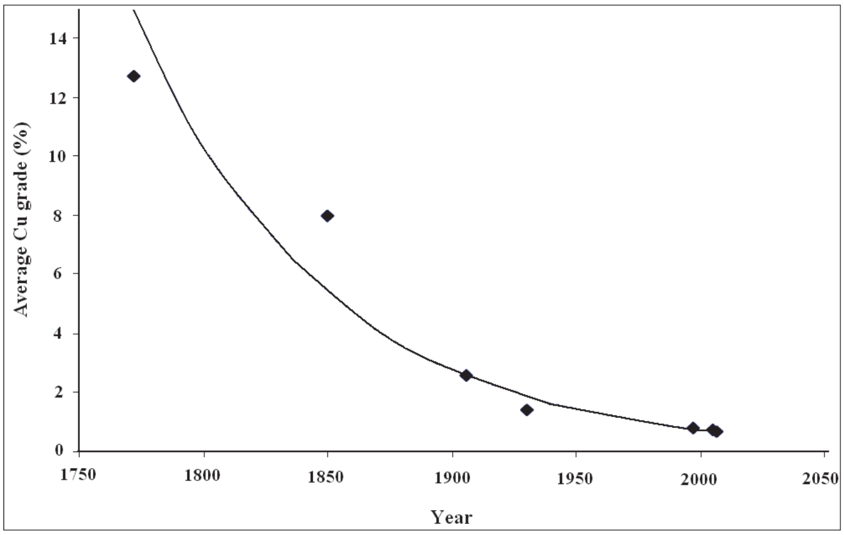


Fig. 1: The copper deposits ore grade during 1770- 2007 [27].

The deposits, for which we now search, are largely concealed by weathered outcrops, drift, soil, or some other covers and, thus, sophisticated exploration methods are required to find them.

The quality of ore is measured by many different factors, typically the ore grade and its distribution, ore body continuity, the workability of natural deposition of the mineralized bodies, and the ore process-ability. Nowadays low grade mineral deposits lead to a very high ore tonnage excavation with the adherent economical and environmental problems. As poorer (low-grade) the ore quality as higher will be the costs of recovery of the valuable products [87].

As a general concept for the cut-off grade within a given ore body, the tonnage and average grade always go in inverse direction: the higher the tonnage, the lower the average grad. Each cut-off grade gives a unique set of tonnage [109], average grade and, hence, the amount of mineral products that can be recovered and those which are lost.

With the lower ore grade, high excavation, haulage and processing expenditures will yield and leave behind increased harmful waste materials. Figure 2 shows the huge equipments for extraction and transportation, which are utilized nowadays for a great extent to compensate for the lower ore grades.

Fig. 2: The huge equipments for extraction and transportation

Surely the environmental and rehabilitation problems, which are engaged with the associated increasing mining and operational costs, will decrease effectively the overall net present value (NPV) and the feasibility through the project life.

1.2.2 World markets

Modern transport and communication lead to many commodities to have a world market. Thus, minerals market prices are sensitive to any change in worldwide supply and demand, that price change in one part of the world affects the price in the rest of the world.

Because of the present rarity of the availability for strong economically mined deposits and the very high operational, maintenance and spare parts, equipments and processing costs, mineral commodities, especially metals and industrial minerals, show an increased trend in their prices.

Figure 3 shows the world gold and silver metal average prices during 1900-2012 [130]. In the same context, the average price of some important metals from 2000 to 2011 is shown also in Table 1 [60, 62].

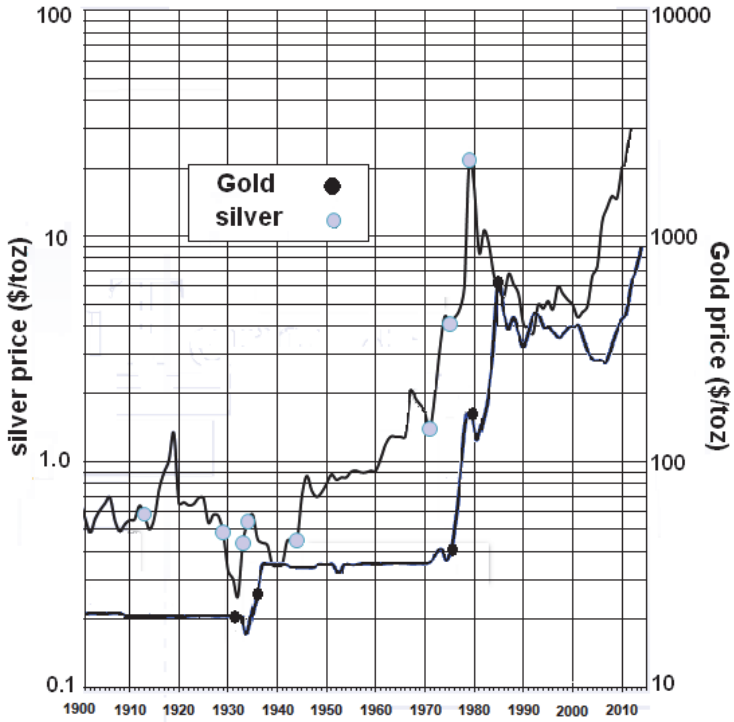


Fig. 3: World gold and silver metal prices during 1900-2012 [130]

The increasing trends of metals prices are obvious. Taking into account the decreasing trends in the new mineral resource availabilities, more intensive capital is also associated with the difficulty to operate the ore bodies, which become of difficult nature, deeper, less accessible and more difficult and expensive to be processed.

Table 1: Average metals price from 2000 to 2011 [60]

Metals	Commodity price from 2000 to 2011				
	Unit	2000	2009	2010	2011
Aluminum	$/t	549	1,665	2,000	2,100
Copper	$/t	1,813	5,150	7,000	7,950
Gold	$/oz	279	973	1,000	1,572
Iron ore	¢/t	28.8	101.0	120.0	169.65
Lead	¢/kg	45	172	225	255
Nickel	$/t	8,638	14,655	17,500	22,420
Silver	¢/oz	500	1,469	1,550	3,015
Tin	¢/kg	544	1,357	1,650	2,500
Zinc	¢/kg	113	166	225	240

1.2.3 Sustainability requirements in mining, environmental and social issues

In 1987, sustainable development has first been defined by the World Commission on Environment Development (WCED) as a development that "*meets the needs of the present without compromising the ability of the future generation to meet their own needs*". The concept requires the integration of economic, environmental and social considerations into all decision makings, taking into account the intra-generational equity through the alleviation of poverty by concentrating the benefits of development in lesser developed areas and considering the needs of the future generation to ensure that inter-generated equity exists. Sustainable development should also ensure uncontaminated environment [109].

Despite undesirable outcomes in the past, the mine planning process continues to focus on technical and financial considerations while environmental and social objectives considered later in the design sequence, unfortunately more often in the form of impact mitigation.

Mine design which maximizes the NPV without the environmental consideration during planning is not really an optimum design [106]. Moreover, the upward trend in restrictive environmental laws and regulations around the world, demonstrate the truth of harder mining operation circumstances. The success of a mining company without the consideration of environmental issues seems very unlikely. On the other hand, postponement of the environmental measures is not reasonable and causes much more costs in future, which act as recovery rates, energy consumption, CO_2 emissions, etc.

1.2.4 The strategic importance of the mining industry

Mining is an industry of a long life cycle, numerous steps and multiple operations, which require huge investments. These steps may be concluded as: *Mineral prospecting & exploration*, *Pre-feasibility & feasibility study*, *Mine development*, *Mining*, *Mineral processing*, *Smelting*, *Refining*, *Marketing*, and finally *Closure*.

Mineral prospecting & exploration, to discover a mineral deposit; *Pre-feasibility & feasibility study*, to prove its commercial and environmental viability; *Mine development*, to establish the entire infrastructure; *Mining*, to extract the ore from the ground; *Mineral processing*, includes milling of the ore, separation of ore minerals from gangue material, separation of the ore minerals into concentrates and separation and refinement of industrial mineral products; *Smelting*, to recover metals from the mineral concentrates; *Refining*, to purify the metal; *Marketing*, for shipping the product (or metal concentrate if not smelted and refined at the mine) to the buyer (custom smelter or manufacturer); and finally *Closure*, that before a mine has reached the end of its life, there should be a management plan for mine closure, which details and costs of the proposed closure strategies including the environmental issues, the costs of employee retrenchment, and social and community implications.

Moreover, it can be expected that many years will elapse between the start of the exploration program and the start of the real mine production, with limited exceptions [28, 124]. During this time gap, no return is being made on the invested capital. Also, the chances of success in exploration, in certain circumstances, may be of a far low percent. Despite such a high element of risk, conscious countries, substantially, capitalize in mining industry.

The successful mining can provide a much higher profitability than that of most other industrial ventures. Practically, achieving the balance between these strategies is the real challenge. Figure 6 shows a schematic of the currently challenges of mining practice.

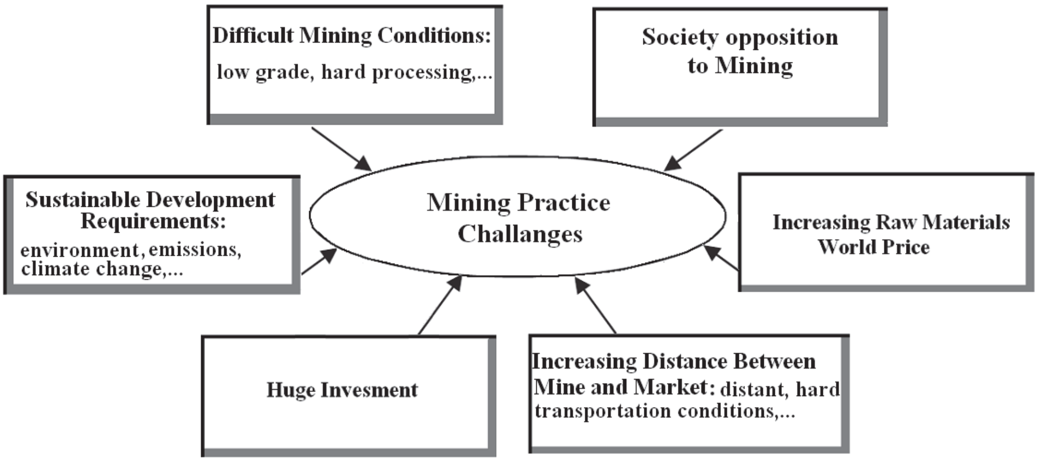


Fig. 4: Schematic of challenges of mining practice.

Because of all of these precedent requirements and challenges, which are reviewed in this chapter, planning optimization through is essential in order to realize the economic benefits of mining and processing of the required ore quantity production with the best quality and the lowest costs. The following section will review the optimization concepts for the overall holistic optimization (Mine-to-Mill optimization), state of science in this subject and the aim of the present work.

2. State of the Science and General Outline for the Mine Planning Optimization Concepts

2.1 The mine planning optimization concepts

2.1.1 Improvements for the interconnected mining and processing operations

Blasting, mining and processing

Generally, for mining, the function of extraction planning and of the drill-blast, loading and hauling operations, is to deliver material to the processing plant. In the past, the primary focus was on the ability of the excavation equipment to productively dig the blasted rock, and on the amount of the oversize produced chunks. Intensive study of the economic relationships between these unit operations and how the choice of technology for each step can affect the overall cost, not only for the delivery to the processing plant but also for the processing operation as well, is relatively new [16].

Blasting is a controlled destruction of the rock mass performed to loosening, fracturing and moving rock fragments with certain specifications to begin the extraction process. Holes are first drilled to specific geometric constraints to accomplish the process. Because of geologic conditions, safety and environmental concerns, holes are not always located at precise locations and in fact the locations are statistical in nature.

Explosives are then loaded into the holes with varying ratios dependent on the water conditions and the equipment considerations, which give more other statistical variables. The timing of blasts, explosives performance parameters, and geologic conditions are other factors. Finally, the blast is fired and the results are evident in the fragmentation degree, movement, and the physical properties of the blasted material.

Thereafter, mining operations begin. Excavation machines, such as power and hydraulic shovels, backhoes, multi-buckets excavator…, dig and excavate the blasted material and load it onto trucks to be transported to the processing plant, which is also a statistically process, as it is affected by the degree of fragmentation and other blast results. Digging rates, cycle times, and the amount of truck fill are related to the blast results.

Mining and blasting variables are interrelated and affect each other significantly as well. For example, the selection of the number of trucks and loaders in the mining cycle can affect the optimum blasting explosives ratios for maximum throughput, and vise versa. Also, particle size distribution and the shape of the final rock product are becoming more important, especially in connection to the subsequent operations, starting with the loading rates and ending with processing. Figure 5 shows schematic of a simplified mining arrangement illustrating the degree of complexity and dependence involved [75].

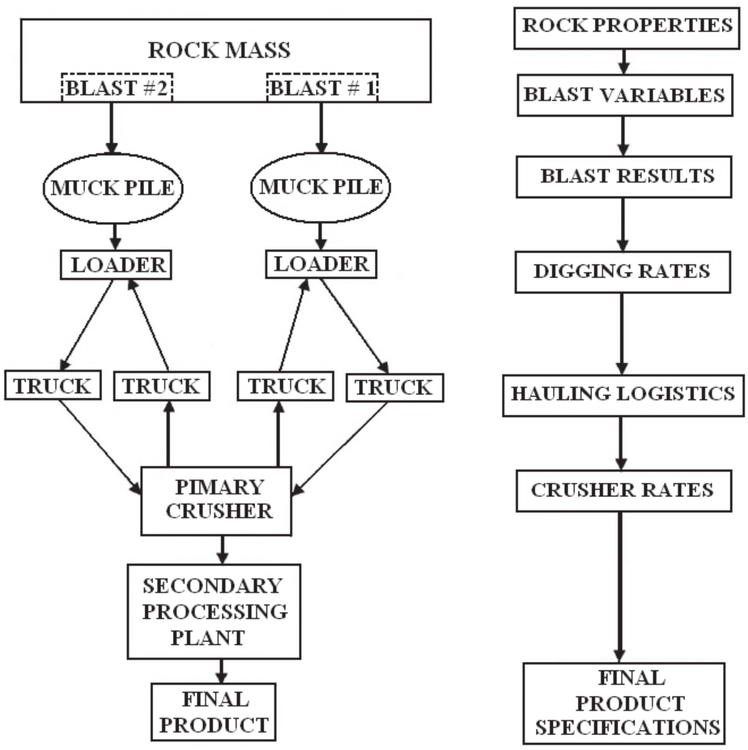


Fig. 5: Schematic of a simplified site-specific mining arrangement [75].

Crushing and grinding are the principal stages which are functioned to reduce the ROM ore to an appropriate size, whether for direct sale or to liberate the valuable minerals for subsequent beneficiation stages such as flotation and leaching. The capacity and efficiency of such size-reduction processes are strongly influenced by the fragmentation distribution which in turn is influenced by blasting. Autogenous (AG) and Semi-autogenous (SAG) milling, for example, rely on the feed ore for grinding media and hence are relatively sensitive to changes in feed size distribution [88]. Hence, a potential exists for modifying the blasting strategy to benefit the subsequent size-reduction stages.

Net present value (NPV) maximization and online relocking

The Net Present Value (NPV) is the standard and the most commonly used criterion, which incorporates a means for dealing with unsteady and uncertain economic conditions. This criterion for any mining operation is the sum of all future cash flows discounted by an appropriate rate of interest, which should at least be the cost of capital [85].

Moreover and from the standpoint of the mineral resource exploitation concepts, the cut-off grade is the grade at which the resource material will meet all the costs associated with its depletion to a marketable product, according to a general plan that defines quantities, costs, and efficiencies over a defined period.

Ores in general are defined operationally by a cut-off grade. Materials with a mineral content above the cut-off grade are scheduled for further processing. Other materials are left, or dumped as waste. An essential preliminary, to an analysis of cut-off grade strategy, is the examination of the net present value (NPV) maximization for an operation, based upon a finite resource [71, 72].

Critically, it should be denoted that the cut-off grade, especially with the valuable metal ores, are not fixed item but it is highly variable, for example annually, with so many factors such as world commodity prices, introduction of new mining and processing technologies,… and so on. Thus according to this, it should be more than one dumping pile, according to its grade, even if it was less than the cut-off grade.

The availability of scheduling software allow mine planners to analyze several scenarios and select the one, which meets the stated criteria, based on various combinations of online re-blocking, grade intervals, number of pits and production constraints. The results may provide valuable information on the critical factors that impact on the cash flow of the mine.

The optimum annual schedules that will give the highest (NPV), while meeting various production, blending, sequencing and environmental constraints, is very important, when planning for an open pit mine, for example.

Customer's identification

Identifying customers is a key part of any successful business. For example, blasting engineers must know who their customer is, if they endeavor to produce the best product at the best price. This customers list may include the followings:

- Shovels have to dig efficiently.
- Crushers must be able to crush without plugging.
- Mine operations must avoid excessive delays and safety problems.
- Neighbors must not be exposed to environmental problems.
- Mills must maximize throughput while minimizing energy consumption.

Failure to achieve any of these goals would critically compromise mining operations.

Unfortunately, the last point, which is to provide maximum mill throughput while minimizing mill energy consumption, is, somewhat, a poorly understood relationship.

Important contributing factors, including geological variation, crusher performance, seasonal temperature variations, plant figures, set-points and operational parameters, and maintenance issues, should be recorded and comprehended. This is important in order to well understand, for example, the effects of blast design and the chosen ore blocks on the mill performance and energy consumption. Therefore, mines have to capitalize on the economic potential of energy optimization.

Advanced quantitative and monitoring methods

The mining and processing practices have significantly advanced through the addition of more quantitative methods including: simulations and modeling of blasts, mine processing and mine equipments; extensive mining production databases; digital evaluation of blast results and fragmentation size analysis; use of GPS, GIS and Laser profiling technology within drilling, excavation, loading, hauling and tracking; computerized crusher control systems; more precise drilling methods; and 2D and 3D rock mass and geochemical modeling.

The number of steps, their complexity, and interactions in most mining and processing operations make trial-and-error attempts at achieving global optimization difficult and expensive. However, modeling and simulation offer a cost-effective and rapid way to a successful outcome, which may acts as increased revenue from a higher varying ratio of ore lump to fines, increased milling rates, or a new heap leach size-distribution, which enhances the valuable minerals recovery.

In some cases, the conditions required for optimizing any one of the mining and processing stages may be counterproductive for the optimization achievement for another downstream one. An approach is therefore required, in which conditions for each step are varied so as to achieve global optimization, which can be called Mine-to-Mill optimization.

2.1.2 Urgent demand for the unit-operations cost reduction through holistic optimization

Mine decision-makers are under heavy pressure to decrease the operating costs, as the cost of supplies and services continue on an upward trend. Most decision-makers have looked at functional improvements to apply technology to replace staff or enhance decision making in somewhat localized areas of the operations.

Historically, the production of a mineral commodity has been perceived as two distinct stages: 1) mining to extract the commodity, and 2) processing to convert it into a marketable end-product. However, mining and processing are intimately linked, particularly on considering of the particle size reduction.

Without considering the entire system, optimizing of each stage separately often looses the better economic and energy-saving opportunities. Mine-to-Mill technology takes the entire system into account, from the blasting, even from the drilling, operation to the plant size-reduction circuit.

Moreover, one important requirement for any mineral deposit to be feasible is that its ore-body should have susceptibility to be processed in an economical manner [94].

Hence, the mine planning optimization goals are to:

- achieve consistency in production,
- reduce process costs,
- improve process control to maintain stability, predictable and stable processes, streamline process flow, and
- consider environmental, social and health requirements.

This optimization will also decrease downtimes and shorten cycle times, focusing primary on the reduction of energy consumption in the production process to maximize benefits and achieve sustainability.

2.1.3 Expenditures of size reduction operations

Costs of the first phase of size reduction (Blasting and Crushing)

Blasting is considered the first phase in the material size reduction. The powder factor (p.f.) is defined as the weigh of explosive required for a rock unit volume to be fragmented to a certain size. Figure 6 shows a simple description of the (p.f.) effects on the costs and the productivity.

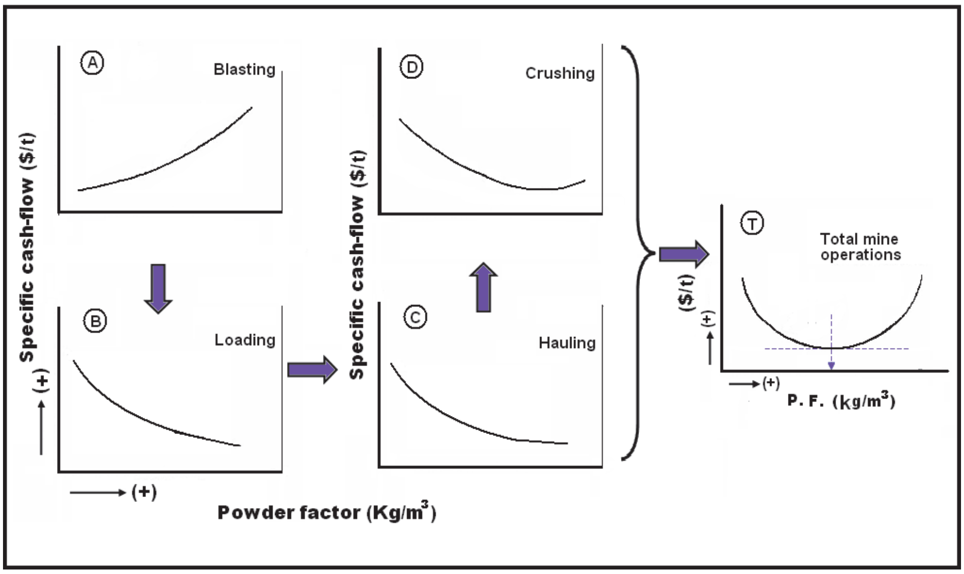


Fig. 6: The powder factor effects on the subsequent operations costs [82].

Improved fragmentation accomplished in blasting (sub-figure A) not only improves digging-ability, loading rates (sub-figure B) and the loading-hauling cycle time (sub-figure C) but also reduces the workload in crushing and grinding operations (sub-figure D).

The maintenance expenditures in the mine and plant are also affected. The maintenance cost of excavator buckets cutting tools, crushers, and mills liners is reduced by improving their wearing rates, as the ore fragmentation are generally finer with the higher powder factors [82,102].

The effect on the primary crushers is mainly by increasing of through-put. While with grinding and milling, which consume the bulk of the electric energy in the ore size-reduction, their costs drop considerably when more are spent on blasting, owing to their more improved feed, which introduced by the crushers, due to the tighter CSS of the crusher.

Costs of the advanced phase of size reduction (Coarse and Fine-grinding)

The energy input to further size reduction is great. The overall size reduction, which is performed in a series of stages, can be from 80 % feed size passing of 40 cm to a final product size of 45 µm (325 #). To accomplish this, a lot of energy is expended, with much of the energy input being dissipated as heat [131].

The third theory of size-reduction developed by Bond [14], at which work index was measured and reported for many rocks, is still used today, although there have been recent advances [113].

The formula of the Bond's third theory of size-reduction is [14]:

$$E_r = \frac{1}{10} \times WI \times [pX^{-0.5} - fX^{-0.5}] \qquad (1)$$

Where: E_r required energy for size reduction, (kWh/t); *WI* work index, (kWh/t); *pX* 80% passing size of the product, (µm); and *fX* 80% passing size of the feed, (µm).

Using this theory, energy requirements to reduce fragments from an 80 % feed size to an 80 % product size can be calculated. Table 2 shows the average values of Bond's work index for various materials, which is the specific energy required to reduce the material particle size from any feed grain size to 100 µm [38]. Thus, one can study the work input required for different feed sizes and work indices in the stages of size-reduction.

Table 2: Average values of Bond's work index for various materials [38].

Material	Bond. index	Material	Bond index	Material	Bond index
Basalt	19	Granite	11	Pyrite	10
Bauxite	10	gypsum (rock)	7	Quartz	15
Cement clinker	15	Hematite	14	Quartzite	11
coal	11	lead ore	13	Rutile ore	14
coke	17	limestone	14	Sandstone	11
corundum	35	Magnesite	12	Quartz sand	16
Dolomite	13	Magnetite	11	Silicon-Carbide	29
Feldspar	12	marble	12	Slag	11
Ferro silicon	11	Zinc phosphate	11	Zinc Ore	12
Flint (rock)	29	Potash	9	Zircon	20
Fluorspar	10				

Table 3 shows the feed and product size, the calculated total energy input, according to Bond's equation, and the energy cost for each unit operation for a specific mine, according to certain p. f. and explosive and electric costs [133]. The explosive cost is based on the (p.f.) of 0.33 kg/t (0.65 lbs/ton) and an explosive cost of \$0.264/kg (\$0.12/lb). Electric energy cost is assumed to be \$0.07 per kWh.

Table 3: Energy and cost calculations by unit operation [133].

Operation	Feed size	Product size	Work input	Energy cost
	cm	cm	kWh/t	\$/t
Explosives	∞	40	0.23	0.087
Primary crushing	40	10.2	0.24	0.016
Secondary crushing	10.2	1.91	0.61	0.043
Grinding	1.91	0.0053	**19.35**	**1.35**
Totals			**20.43**	**1.50**

Within another case, summarized in Table 4, the operating cost of a SAG mill circuit at a certain mine [56], is calculated. The mill power consumption averages 9.3 kWh/t and the recycle crushers 0.45 kWh/t. Ball consumption is typically 0.54 kg/t.

Table 4: SAG mill circuit major operating cost [56].

Operating Expense	**OPEX $/t**
Power	
SAG Mill	0.34
Recycle Crushers	0.02
Grinding Media	
SAG Mill	0.52
Crusher and mill Liners	
SAG Mill	0.17
Recycle Crushers	0.03
Total SAG Circuit	**1.08**

These costs alone account for 33 % of the total concentrator operating expenditure. The average annual processing cost is 3.29 $/t. The entire grinding circuit acts up to 70 % of the total processing cost.

Since metal mines ROM may subsequently milled to finer than 25 µm (500 #), significant cost reductions can be documented, as tens of millions of dollars annually, through total flow-sheet optimization. This optimization, as Mine-to-Mill concept, will provide enormous unrealized cost and productivity improvements to metal producers.

2.1.4 The Mill as a critical point in the product supply chain

Significant benefits will result if a broader scale approach is applied, reaching from the Geologist to the Metallurgist, i.e. incorporating the various disciplines of geology, mining, metallurgy and engineering, to optimize the entire plant size-reduction circuit.

The mining engineer should be able to monitor, online, “what” is being delivered to the primary crusher to better understand the size distribution of the ROM feed and trace the effects of changes in blasting and mining parameters that affect fragmentation and, consequently ultimately, affect the mill throughput. Also, the relatively low grade of the ore deposit requires the mine to be extremely efficient in all aspects of the operation by increasing the milled tonnages and recoveries in order to maintain the profitability [100].

The objective of introducing Mine-to-Mill concepts is to tie the collection and use of data to the product supply chain (the ore flow process) at critical points, thus building the connection to the customer (downstream operator).

Locking forward to a Value Adding Systems, one of the first steps is to model the metal supply chain. This Model should detail the physical process beginning from geology models and starting with a high level block flow diagram. Each of these blocks should be decomposed into detailed models, which in many cases already exist. The modeling process should focus on the detailed information required to flow between the blocks in order for the downstream process to improve its operation with further connections, on a reverse basis, to provide feedbacks. Figure 7 shows a schematic for technical parameter control loop with the supported flow data [39].

The integration of the processes from geology to mill must be investigated particularly by using information technology to provide accurate ore quality information to the mineral processing operators. In this closed loop of knowledge management, mill feed fluctuation can be decreased and ore-predictability increased, focusing on increasing the quality and quantity of product produced by the whole operation, not just the mine or mill separately. A hard look should be taken at the total process and how it works together, not just the functional parts in isolation.

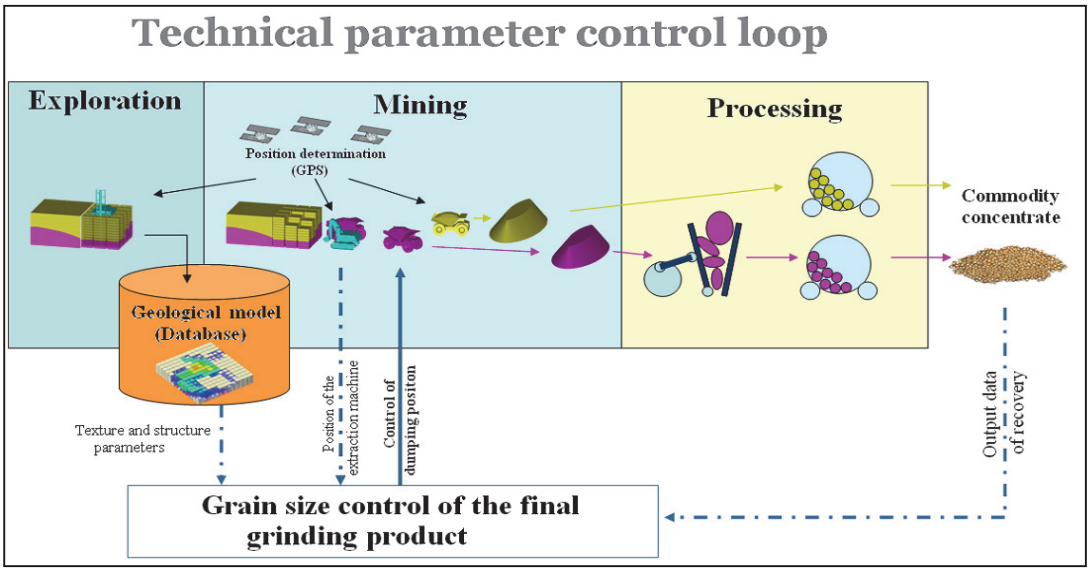


Fig. 7: Schematic for a technical parameter control loop [39].

2.2 Critical review of researches for the (Mine-to-Mill) optimization field

The researches in the (Mine-to-Mill) optimization field move in three directions:

- Mill throughput optimization
- Intelligent assistant systems and processes automation and monitoring
- Scheduling software and operationally holistic modules

2.2.1 Mill throughput optimization

An intensive survey indicated that the efforts to date in Mine-to-Mill optimization have been concentrated primarily on optimizing the relationship between blasting and the plant size-reduction. Many studies are conducted in order to help in introducing a controlled and an appropriated feed size to the mill stage, primarily to increase mill throughput.

These studies, commonly, are based on changing and designing of different blasting strategies. These include blasting and drilling parameters manipulation; during- and post-blasting ore monitoring, by digital size analysis using photos, videos and image processing softwares; face profiling to minimize shot muck dilution; and dividing the in-situ ore into different certain categories each with its own blasting pattern, [3, 17, 45, 46, 50, 52, 56, 61, 65, 89, 126, 133].

But these researches are lacking in investigation of the other mining operations and their probable effects on the optimality of the ore delivered to the mill, such as drilling, stockpiling and primary crushing.

One of the factors that limited the usefulness of blast fragmentation models is the lack of information on the in situ characteristics of the rock mass. This information includes rock strength, RQD (Rock Quality Designation), fracture spacing, and fracture orientation. In most mining environments these parameters can be highly variable.

The drill monitoring data, not only the drill core itself, which may help in obtaining some of the rock mass properties, is not intensively investigated. If considered well, it may present a geologic-unit specific combination of blasting, excavation and transportation efficiencies balanced with milling capacities and liberation characteristics.

It is noted also that all of these optimization trials are concentrated on the ROM feed size and neglected other important physical properties such as hardness, texture and internal fractures, which require further investigations.

2.2.2 Intelligent assistant systems and processes automation and monitoring

Many efforts are exerted in order to use the intelligent assistant systems and processes automation to increase productivity and improve utilization of the individual components capacity. These include infrared sensor technology [95], GPS, Radar, LIDAR (Light Detection and ranging), odor detectors [94] and optical processing systems [104].

More automation and monitoring documents can be reviewed in [44, 86, 109, 123]. With the combination of these technologies, an efficient system can be available to the mining industry to online-monitoring, tagging and tracking the equipments and ore mass flow.

But however, in order for these assistant systems, to be able to facilitate the knowledge held in the mining and geological models to be transferred to the mill control system, these disjointed controlled individual systems should be collected and assembled to one holistic system.

Although the desired integration is difficult, due to the high variety and, to some extent, inhomogeneity of the measured operating parameters between the various stages of mining and processing operations, it is essential, in order to have an optimal performance. Moreover, the success in cost reduction and increasing productivity will be engaged with the continuity and sustainability in mining and processing.

Also, a change in the planning approach should be existed, in order to consider, support and invest in the high quality functional and integrated systems to realize and insure sustainability and environmental protection beside profits.

2.2.3 Scheduling software and operationally holistic modules

Some researches are directed towards modification of the cut-off grade models, which relates the net present value with mine and plant operations and constrains, [31, 37, 98, 99, 107, 129]. They could realize improvements on NPV, but just with the given ore hypothetical conditions.

The development of scheduling software that considered the effects of re-blocking and stockpile intervals on the mining schedule and material movement is also reviewed [24]. In this document, it is recommended that a smaller interval, specified with the re-blocking and stockpiling, is better for the positive effect on the scheduled mill grade. For more about mining scheduling, [31].

Other researches are directed to developing models to predict the fragmentation due to blasting [2, 52, 63, 103], such as the image processing software, for assessing post-blast fragmentation and predict crushability of the ROM, engaged with blasting charts and models to predict fragmentation before blasting. Data are collected and analyzed for each blast and by continually updating of this database, as mining progresses, accurate model, with time, can be attained. For further researches on modeling and integration of mining and processing stages, [3, 16, 53, 65, 75, 76, 78, 79, 101].

As a common trend, these models are focusing and dealing with just one or, at the best estimation, two steps in the large production chain of the mining industry.

In the context of trials to realize the Mine-to-Mill holistic modules, the most recent and promise is represented in [40], where an innovative methodology is developed, that provides an adaptation between different grain size ore stockpiles of definite ore blocks and a circuit of designated mill-lines.

This approach, theoretically, resulted in a higher efficiency in milling capacities utilization and a reduced energy consumption costs. Therefore, it is preferred here to be a starting point in the current study, in order to improve and develop an approach that may help to make it practically realized and generalized to achieve an overall-controlled and optimized mining and processing planning.

From the previous review on the efforts done, in order to realize improvements in the field of mining and processing, especially for the metal mineralization, areas of development possibilities can be determined. This area, which opens new concepts for further investigation in this thesis, is mainly inspired from [40]; and can be concluded through following shortness:

- The method considered just one-metal ore, while many metal deposits, in fact, are multi-metal ore deposits.
- The method denoted the relations between the various mining and processing stages (until milling stage) and did not investigate deeply the related natural and operational parameters that may interact between these stages.
- In order to be of valuable practicability, these interrelated modules need to transfer into real flowcharts that consider multi-metal ore deposits with different physical and mineralogical properties.
- It considered just the ore grain size, as an absolute constraint, in the milling stage and neglected other ore physical and geological properties, which may have a considerable effect on the milling efficiency and energy consumption, such as hardness, texture and internal fractures.
- Some boundary activities are neglected or not sufficiently investigated such as:
    - The drilling parameters monitoring and their probable effects on the subsequent decisions; and
    - The suitability of the milled final product for the subsequent refining separation such as floatation.

2.3 The aim of work and the thesis layout

The Aim of Work in is assigned from the following notes:

- Mining and processing operations involve a variety of steps, each of them has its own characteristics and requirements for efficiency. Therefore, a holistic approach is essential, in which the conditions for each step can be adapted to achieve global optimization.
- Especially with surface mining, a major problem exists, that most mines have several blasts and excavators in various locations, which may have varying geological and petro-physical groups, thus results can be different when treated in one direction.
- A comprehensive mineralogical investigation will allow the prediction of factors, such as grind size for effective liberation, texture and hardness, maximum grade attainable, and the probable extent of tailings losses. This will provide high quality database to be used in mill optimization.
- Also, the investigation of metallurgical properties and tracking the ore mass flow from stope to the mill is essential to challenge the variability in the ore feed, in order to facilitate recovery improvement and reduce consumable and power costs.

Therefore, the aim of work can be concluded in designing of an approach for the global optimization of the integrated mining and processing operations, through the mining selectivity strategy, the deeply investigation of the ore deposit parameters, and the proper adaptation and planning for the plant facilities. In the current work this will be done mainly through a dynamic modeling and simulation for the whole mining and processing sub-operations.

The Thesis Layout*:*

The thesis is consisting of seven chapters, (Fig. 8):

Chapter one includes an introduction with a statement for the justification and importance of the mine planning optimization through clearing for the urgent need for general optimization in the mining industry.

Chapter two includes state of the science and general outline for the mine planning optimization concepts, review of researches, area for further development possibilities, and the aim of work.

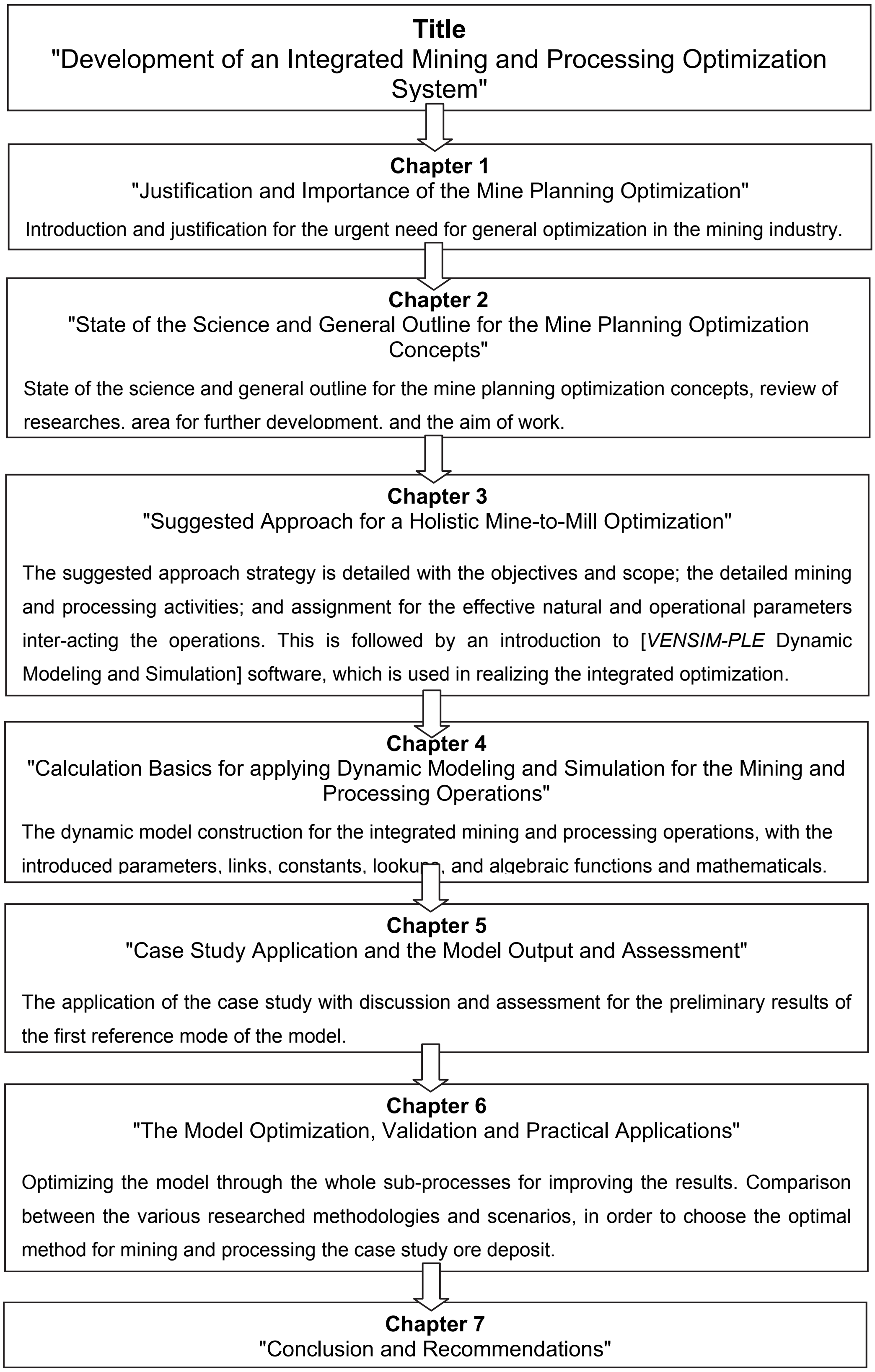


Fig. 8: The thesis constituents and layout.

Chapter three includes the suggested approach for a holistic mine-to-mill optimization, the methodology strategy, presenting for the detailed mining and processing sub-operations, and assignment for the effective natural and operational parameters interacting the integrated optimization. This is followed by an introduction to the modeling software, which is intended to be used in the thesis for tracing and realizing the integrated mining and processing optimization.

Chapter four includes the procedure details for the dynamic model construction by the [*VENSIM-PLE* Dynamic Modeling and Simulation] software for the integrated mining and processing operations, which includes the calculation basics and the introduced parameters, links, constants, lookups, and algebraic functions and mathematicals.

Chapter five includes the case study application with discussion and assessment for the preliminary results of the first reference mode of the model.

Chapter six includes the model optimization, validation and the practical applications, which will be done mainly by optimizing the model (reference) mode to a (controlled) one, in order to generalize the optimization through the whole sub-processes and improving the results. This will also includes comparison between the various researched strategies and scenarios, in order to choose the optimal strategy for mining and processing the ore deposit, which is introduced to the model through the investigated case study.

Chapter seven includes the conclusion and recommendations.

3. Suggested Approach for a Holistic Mine-to-Mill Optimization

3.1 Introduction and scope

Introduction

Here in this study, the suggested approach will represent most of the mining and processing stages. The flexible operations, in which suitable modifications can be made, in order to achieve optimization, will be addressed. In addition, these operations that are affected greatly with the previous ones or have certain influences on the subsequent operations will be investigated in detail.

The main target is to achieve the final mineral product (here, is a metal) with the maximum possible quality, for a predefined or requisite quantity, with the minimum overall production costs, considering environmental and sustainability concepts.

As stated in CH. 2, grinding is considered the most energy consumable stage along allover the mining and processing operations. Also most chances for cost reduction and recovery enhancements are accumulated in this stage.

The main problem within grinding is that the feed of the mills are often variable in its physical properties. This variability is may be due to considering the *average value* of the different ore-body localities as a rule, when dealing with the different variable ore natural parameters as mineral grain size, hardness, texture, mineral content...and so on.

On dealing with the mill feed as a fixed average input grain-size, without special investigations, such as for the internal-fracture, microscopic texture and the different liberation grain-sizes, many subsequent technical errors may arise. The most important from these technical errors is that, if this physically inhomogeneous feed is ground to a fixed final product grain-size, parts of the ore will be over-ground, consuming more wasted energy, and other parts will be not completely exposed, reducing the final product recovery considerably by transfer an amount of the valuable mineral to the tailings.

Therefore, optimization targets and parameters assignment will be focused, naturally, around their predicted effects on the medium and fine grinding processes.

Research Scope

Some boundary stages in the beginning and in the end of the overall mining industry chain will be out of our study, even for their inflexibility to be modified or for their necessity to be achieved as it is. From these stages, for example, are the geological activities before drilling for blasting, however, the exploration data will be studied as input data for the subsequent drilling operation and other next steps.

Another stage that will be out of our scope is the refining stage, as smelting (with metal products), for example, as a result of its inflexibility.

The suitability of the final ground products for the subsequent technological separation methods, such as floatation, will be, rather, investigated due to its great importance and high impacts on the final marketable product.

3.2 The methodology plan

The suggested methodology for a holistic mining and processing optimization is consisting of five main sections:

1. Statement and assignment of the effective inter-related natural and operational parameters interacting the integrated mining and processing optimization. The natural parameters, which belong to the excavated ore and rock description and characterization, are transferred to Appendix 1. The second part, which belongs to the main mining and processing operations, is investigated in this chapter with some technical detailing for their characteristic features. This will help to assigning and selecting the most vital, flexible and effective parameters which have impacts on their own operations and/or the other downstream one(s), especially grinding.

2. Constructing of a dynamic model by the simulating of the global mining and processing operations from drilling to fine grinding, according to the philosophy of dealing with them as one coherent system with functional links and mathematicals.

   The assigned factors and parameters, from the 1st section, will form the main inputs, with others, in order to define the control factors and set points for the optimization along the operations.
3. Experiment the model results by applying a case study practical data and through determination of the optimal blast fragmentation size, the choice of the suitable loading and hauling fleet, and assignment of the corresponding mine life.
4. Optimization of the constructed model will be made by introducing time-based and financial factors with different mining and processing selectivity scenarios, according to certain features and strategies (parallel and series plant arrangements), which will be then identified and described in detail, in order to examine, judge and refine the model results.
5. Afterwards, a mass-flowchart for the suggested methodology will be constructed, in order to conclude the final applicable benefits of the optimization. Also a choice for the best operating strategy will be done, according to the application of the selective mining and the selective (arranged) processing.

A simplified flowchart for the methodology steps is illustrated in Figure 9, while the plant arrangements suggestions due to the planed selective mining and mixing scenarios are shown in Figure 10. The practical expectations from this integrated mining and processing optimization methodology are mainly the overall operational cost reduction accompanied by improvements to the mine product recovery and the environmental impacts by:

a) Right choosing of the blocks to be extracted according to the plant production need.
b) Good utilization of the transporting machines by high adaptation between haulage and extraction operation.
c) Overall energy expenditure reduction, especially in the grinding stage.
d) Ore delivery to plant with high suitability of the physical properties (size and hardness).
e) Good design of the plant production lines in order to be highly compatible with the whole ore deposit properties.
f) Realization of the continuous mining and processing with less environmental impacts.

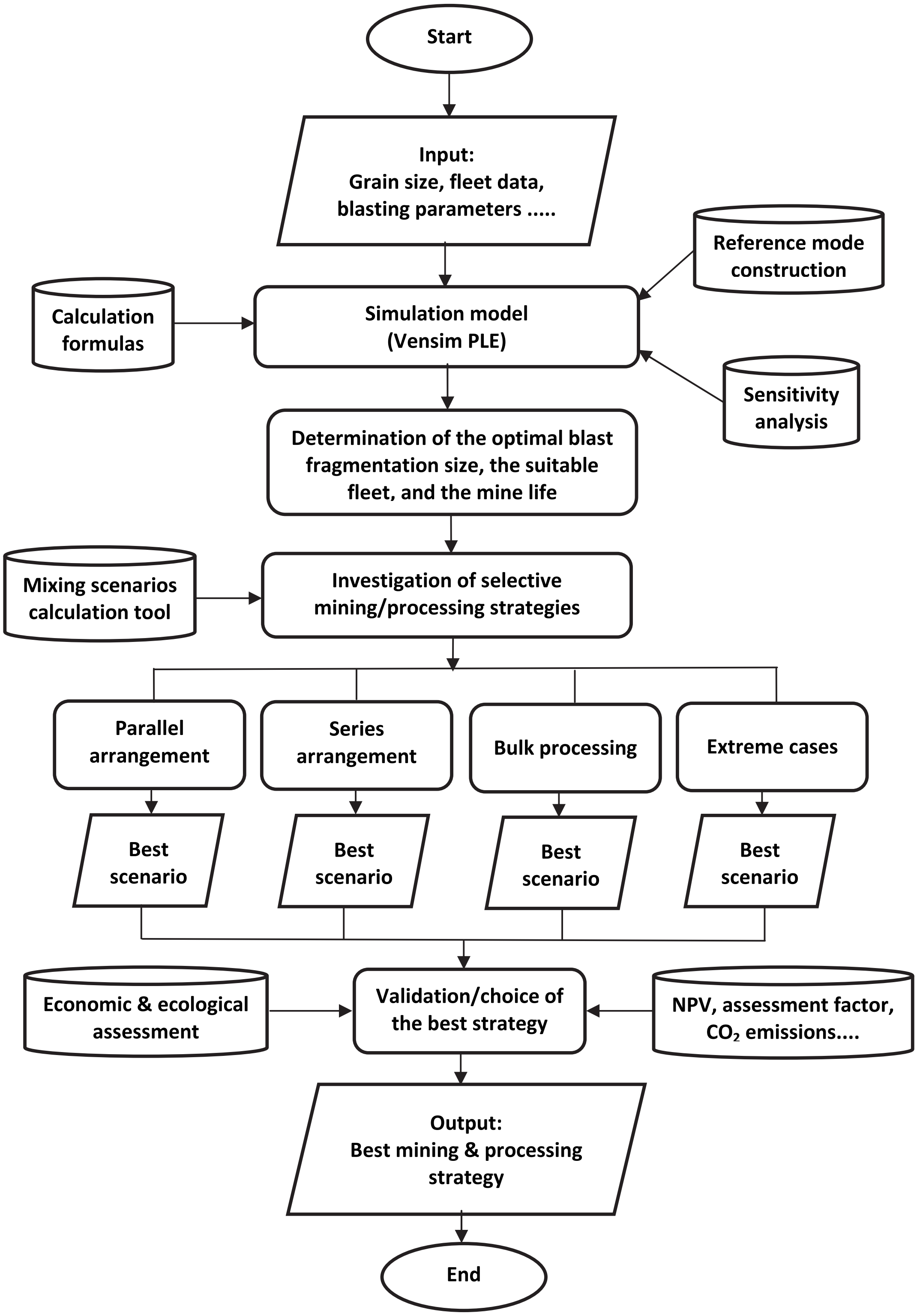


Fig. 9: Simplified flowchart for the methodology steps.

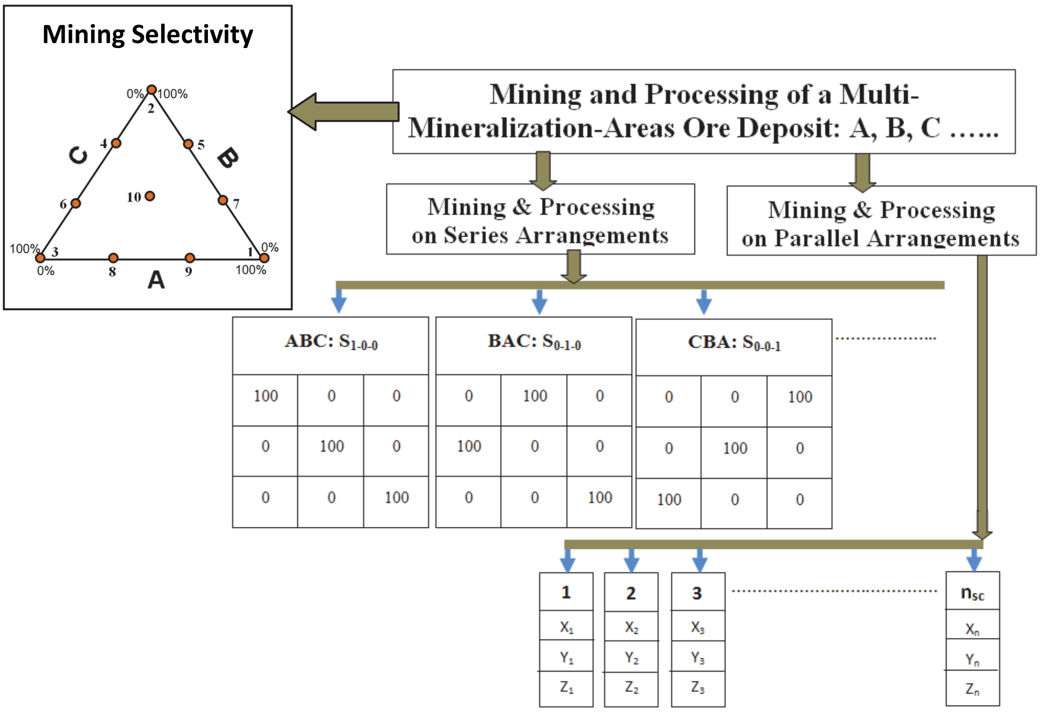


Fig. 10: Plant arrangements suggestions due to the planed selective mining and mixing scenarios.

3.3 Assignment of the operational parameters inter-acting the integrated optimization

3.3.1 Mining and processing activities

The classic main mining and processing operations are presented here with mention to the transient points that connecting in between these two large partials. Figure 13 shows the process chart for the operations flow of mining and processing with the transient activities (dashed lines).

Mining Main Mass Flow Operations

As shown from Figure 11, the main mining activities, including a number of dependent and independent stages, are starting after intensive pre-feasibility and feasibility studies during the mineral exploration. These geological, geophysical, geochemical, mineralogical and economical studies are essential to recognize the main characteristics of the deposit and to deduce its viability. After a time period, which ranges between months to many years, the mining method is assigned, the mine is developed and the production starts.

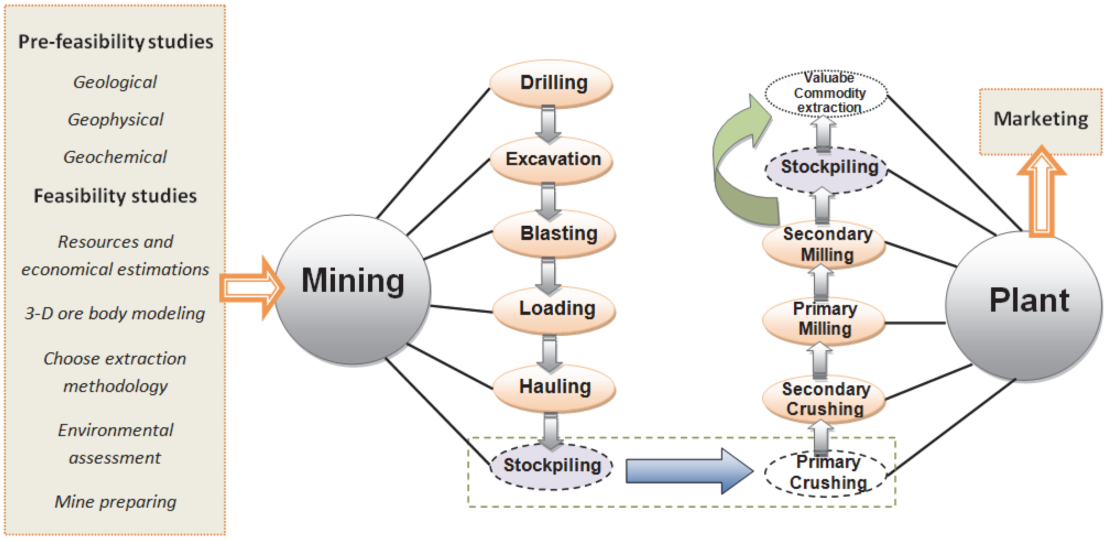


Fig. 11: Process flow chart for the classic operations of mining and processing.

The real first stage in mining and processing is the drilling operation, as a part of the ore deposit fracturing and loosening by blasting. Following to an already planed scheduling, the ore deposit blocks are chosen and drilled carefully to designed patterns, according to their geological and physical features. These boreholes are then charged by explosive materials for pre-defined quantities then fired and blasted, to some extent, to an acceptable fragmentation suitable to be excavated.

Excavation machines, such as power and hydraulic shovels, backhoes, multi-buckets excavator…etc, are commonly used in order to excavate the fragmented material and load it into the hauling equipments. Haulage operation within surface mining is often made by hydraulic trucks. Other transporting equipments, such as belt conveyors, can be used as well, according to many factors such as type and size of the ore, hauling distance, terrain …etc. The transported ore can be, then, stockpiled in stock yards or huge bins or it can be introduced directly to the first stage in the ore processing operations (primary crushing).

Processing main mass flow operations

As shown in the previously discussed Figure 11, the ROM mass flow can be stockpiled or immediately directed to the first stage of ore processing. This takes place, classically, according to many factors, from which the most important is the plant or, to more focusing, the size-reduction plant facilities capacity.

As the ore grain size reaches that size, at which the valuable minerals can be liberated, begins another completely different stage, which is the refining. This stage begins with a special separation procedure such as floatation, heavy media separation, flocculation or ion exchange ending with smelting, in case of metal products, or drying to final products with others.

3.3.2 Mining and processing operational parameters

The diagram in Figure 12 shows the operations flow with the main effective natural factors, which should be recognized clearly and exchanged as a characterized data along the production flow chart, (Appendix 1).

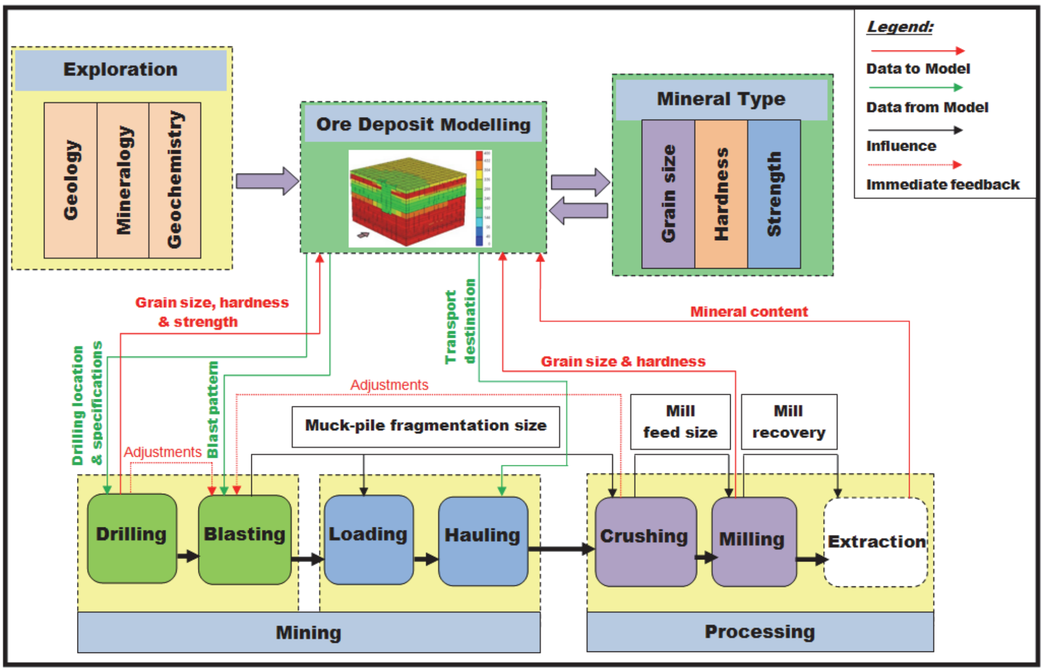


Fig. 12: Mining and processing operations flow and data exchange.

Actually, the operational parameters can be adjusted and manipulated to be compatible with the natural parameters, while the later ones can be just organized after being thoroughly investigated and determined.

In order to be objective and substantive, the operations are organized in the figure in complexes, according to their direct inter-relation or logic consequence, such as two complexes within mining, one includes drilling and blasting operations, and the other includes loading and hauling, and one complex within processing operations which includes crushing, milling and extraction.

In the followings, the main mining and processing stages will be presented and explained, from the point of view of the possibility of the operational parameters assignment, through the upper mentioned complexes.

Drilling and Blasting

The first and the most important real opportunity for collection and identification of the natural parameters characterizing the ore deposits is during drilling operations. Identifying rock natural characteristics and characterization of underlying different layers of rocks can be performed by drilling a series of exploration boreholes and achieved from borehole investigations. As well, the blasting boreholes can be further and more reliable indicators for these purposes, before using it in the blasting operation.

Because drill-monitoring data is available from every blast hole, it provides data throughout the entire rock mass to be blasted.

As a part of bedrock description, a drilling parameter recorder (DPR) system will provide continuous monitoring of the drilling performance [12]. Drilling operational parameters recorders are computerized systems which monitor a series of transducers installed on the conventional drilling equipment to collect data automatically on:

- drilling advance rate,
- down-thrust and pull-up pressures,
- rod torque,
- rotation rate,
- mud/water pressure and flow, and
- depth and time.

The data are displayed, in real time, in digital form and as hard copy, Figure 13, and can be stored on an electronic medium for further analysis (mine planning software).

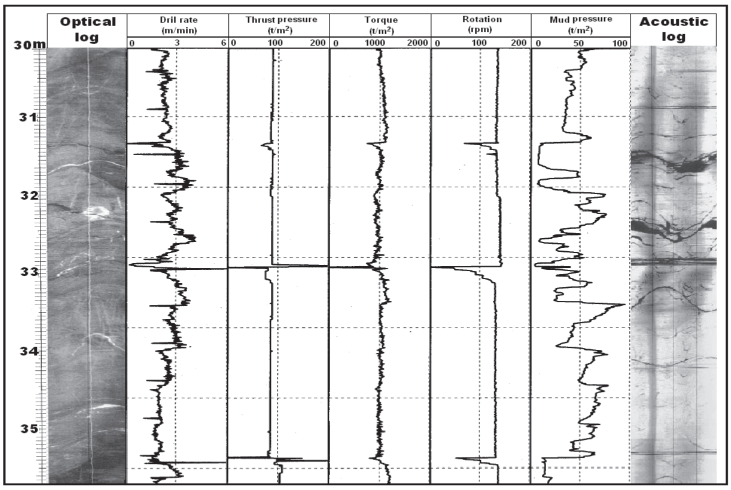


Fig. 13: Real-time drilling monitoring with optical and acoustic logging [12].

If these systems could be also provided with GPS devices, for example, that can be fixed on the drilling rig to monitor the rig condition and location online, this data could be transferred on time to the other downstream operations.

The most subsequent stage that will benefit the drilling monitoring data is the blasting operation itself, by further manipulation of the overall powder factor and stemming ratio or even the blasting pattern as well. The other important operations that will be qualitatively affected by these data are the primary crushing and the stocking, before plant operations.

Drilling parameter monitoring systems help also in evaluation of the natural fractures and joints for the different drilled rocks and formations. This can also transfer a good imagination for the upper-size of the boulders of the resulted blast fragmentation. Of course when this data can be recognized before blasting, and then adjusted properly, it will have a considerable effects on the excavation (loading) performance, hauling capacity and also, specially, on the crushing stages.

After drilling, the carefully logged drilling cores can summarize the following important data:

- lithologic variability,
- fractures attitude, spacing, and types.

Reliable sampling and good investigation of the different lithology from the drill cores can also provide important information about variation of strength, grindability index, texture, crystallography and natural grain size, hardness, moisture content, void ratio… along the vertical profile of the drilling direction. These data can assist energy consumption predictions within size-reduction process, which is considered the most technically and economically critical operation.

Belonging to the blasting operation, it was also realized that the powder factor (p.f.) is an effective economic parameter in this operation. If it is properly adjusted, it may lead to high positive impacts on the subsequent operations. Not only those which are belonging to the mining sub-operations but also the later ones belonging to processing as primary crushing and grinding, can be positively affected.

Improved fragmentation is the core in improving of the blasting operations, which is mainly achieved through the blasting pattern modification and the powder factor manipulation. The powder factor can decide the relationship between the muck-pile handling and its further preparation costs. Its selection should consider the downstream costs in both the mine and plant.

The maintenance expenditures in the mine and plant are also affected. In general, the explosives and blasting cost, has remained roughly the same over the past few years [45], while drill bits, shovel wear parts, crusher liners, labor and electricity have increased. Moreover, the energy consumption by the milling process can be reduced, due to the (p.f.) increasing, through higher throughput and better primary crushing feed to the mills. However, manipulating of the (p.f.) should be achieved carefully due to the following concepts in Figure 14.

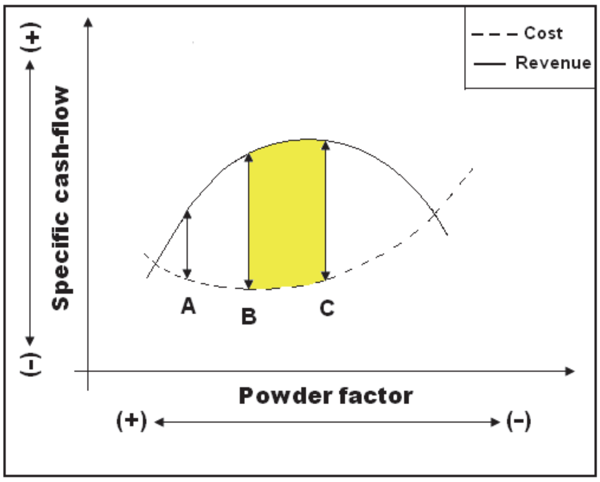


Fig. 14: Allowable range for the powder factor improvements [45].

It is which illustrating the limits of (p.f.) adjustments such as:

- At point (A), a relatively high (p.f.), the dig-ability results will be excellent and loading rates increase. But in the same time, crusher productivity may suffer from slowing and consuming more energy, due to increasing of fineness.
- Point (B) is the (p.f.) for least cost, which yields highest crusher productivity by feeding medium fragments.
- Above Point (C), costs begin to increase owing to creating too large boulders, which hinder the digging equipments and slow the crusher significantly.

Therefore, the most preferable area to get good results is that between points (B) and (C).

Loading and Hauling

The fragmented ore handling system is composed of excavation (loading), hauling and dumping subsystems. The transport of material from production faces to dumping sites is accomplished by rail, truck, belt conveyor or hydraulic transport. Shovel-truck systems are most common in the open pit mining.

Two available techniques, linear programming and queuing models, are mainly used to analyze these systems. The capacity of the used equipments is an important factor and optimization of the equipments combination is essential to maximize availability and, hence, productivity. Studies conducted for the truck allocation were carried out by several authors [92, 1, 114], such as studying the truck-shovel modeling systems as a closed queuing network, discussion of the truck dispatching evaluation using linear programming and studying of automated system for haulage control of the trucks.

In a shovel-truck model, trucks cycle between their assigned shovels and dumps or crushers, over haul roads. When calculating the cycle time for a truck, the variability in time, taken to spot and load, haul, dump and return is considered exponentially distributed and the cyclic queue, is considered to consist of four time phases, (Fig. 15, 16).

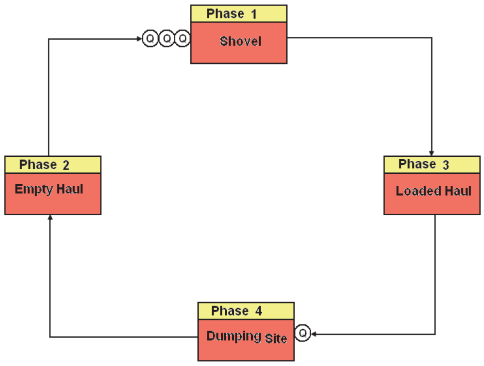


Fig. 15: Main phases of the cyclic.

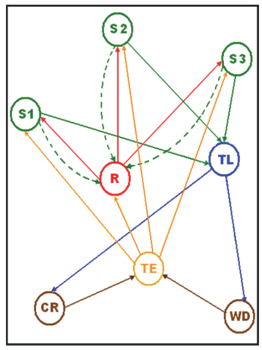


Fig. 16: Probabilities of truck-shovel destination.

S1, S2, S3: Shovels; TL: Truck travelling loaded; TE: Truck travelling empty; WD: Waste dump; CR: Crusher and R: Repair center.

The shovel-truck cycle time phases are as the following:

- The shovel phase, (event: the shovels are loading the trucks)
- The loaded haulage road phase, (event: the trucks are travelling loaded)
- The dump site phase, (event: the trucks are dumping)
- The empty haulage road phase, (event: the trucks are travelling empty).

By reacting and analyzing these cyclic times with the excavator type and its service (loading) rates, production over a given time period can be calculated by the number of loads that trucks take to the dump, or directly to the crusher.

In order to realize truck-hovel matching, according to the plant requirement, starting from the crusher capacity and occupational availability, several areas that affect the system should be identified such as:

- hauling road conditions (altitude, grade and rolling resistance, type of soil, weather conditions),
- the main control room strategy, and
- the dispatching program and data management.

These investigations are important in establishing control parameters for the haul fleet, since time spent in queuing (at shovel, dump areas or crushers) or in correcting of fault orientation is considered as a tonnage lost.

According to the inter-arrival time, frequency and the planed management system, the fleet is processed by the suitable ranking queue rule [92] as:

- FIFO - Trucks are loaded on a (first-in first-out) basis, or
- Priority – Trucks are loaded on the basis of their service requirements.

The last strategy is better for the valuable ore deposits, if planed properly.

To prevent the delay in trucks or idle of the excavator, the number of trucks in the queue, on the hauling roads and those being served must be, online, managed and oriented to realize the steady state in the system. Hence, effective handling systems can be achieved, regarding the received requirement messages from the plant and the number of dumping locations.

Crushing and Grinding

Primary crushing is considered the main intermediate, which links the mining activities results (mining output) and the starting of the plant activities for size reduction of the ROM (plant input).

Primary crushing equipments, Figure 17, such as jaw, gyratory and cone crushers; horizontal shaft and vertical shaft impactors, could be established in the mining areas or in the processing plant according to:

- the long-term mine planning,
- feasibility studies, and/or
- the type and nature of the product.

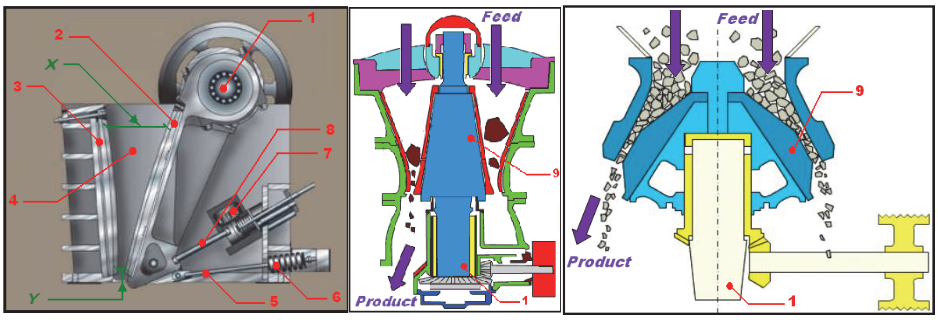


Fig. 17: Scheme for a jaw, gyratory and cone crushers.

1) Eccentric shaft; 2) Swing jaw; 3) fixed jaw; 4) Check plate; 5) Tension rod; 6) Spring; 7) Adjustment chime; 8) Toggle plate; 9) Mantle; *X*) Open-side setting and *Y*) Closed-side setting.

The main importance for the primary crushers is that their products are the feed for the subsequent mills.

The main factors affecting the primary crusher are the natural parameters of the ROM, while its own operating parameters, which can have considerable effects, are only the speed and the closed-side setting (CSS) dimensions. Other operational parameters, considering the primary crushing circuit, are the ore feeding rate (e.g. apron feeding machine) and the circuit configuration.

Crushing circuit configurations could be closed or open-circuit crushing. Jaw crushers are often utilized in open circuits, while cone and gyratory crushers are utilized often in closed circuits which contain classifiers such as screens, if they are used as secondary crushers or as pebble crushers combined with semi-autogenous (SAG) mills, as shown in Figure 18. Advantages and disadvantages of each configuration are dependent on the subsequent grinding strategy, which is dependant on the product type and the required final degree of fineness.

Primary crusher could be subjected to adjustments during operation, to control the loading and avoid the risk of breakdown. For example, the feed rate, the crusher motor speed and also the CSS, if controlled by hydraulic cylinders, can be adjusted on operating.

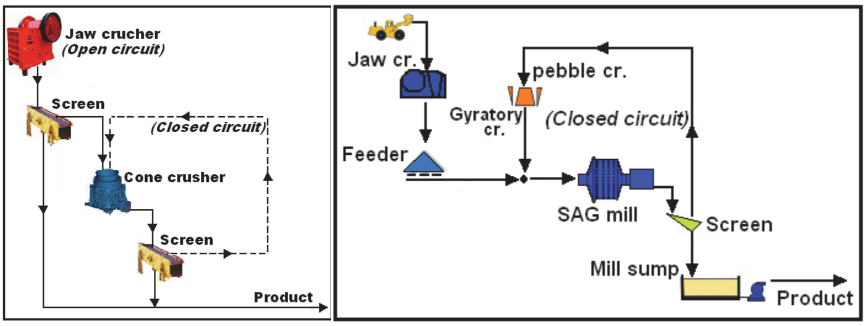


Fig. 18: Crushing circuit configuration, closed or open-circuits.

The main problems, in fact, appear if the subsequent mill is subjected to these naturally fluctuations of the ore to be ground during its operation, taking into account that crushing stage reduces the ROM by only a factor between 4 and 5, while grinding requirements may be reach to size reduction factor between 400 and 500 [46], especially with the metal minerals which are often the final products of ball mills, (Fig. 19).

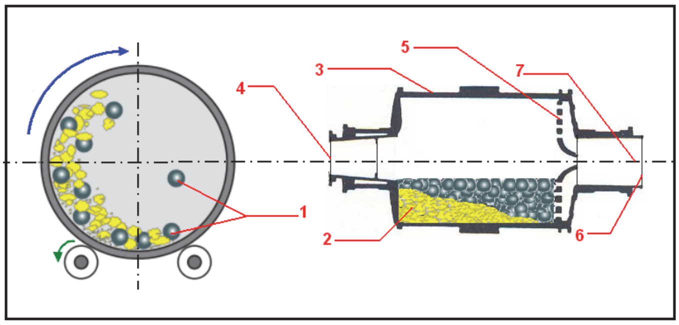


Fig.19: Scheme for a ball mill.

1) Balls; 2) Ore; 3) Mill shell; 4) Feed; 5) Grates; 6) Discharge and 7) Rotation axis.

On dealing with the feed to the mill as a fixed average input grain-size, without special investigations, such as the internal-fracture, microscopic texture and the different liberation grain-sizes, many subsequent technical errors may arise. The most important from these technical errors is that, if this physically inhomogeneous feed is ground to a fixed final product grain-size, parts of the ore will be over-ground, consuming more waste energy, and other parts will be not completely exposed, reducing the final product recovery considerably by transfer an amount of the valuable mineral to the tailings.

The mill operating parameters manipulation and adaptations are more difficult, complicated and time consumable, in comparison with the case of crushers. Changing the grinding media load (e.g. no. of balls, rods...), the charge load (including ore and water, in case of wet grinding) and/or the lining type requires the stoppage of the mill completely for a relatively long times.

The mill speed, for example, is considered a very important operating parameter and it is necessary to be chosen carefully. The mill rotational speed, combined to loading degree of the mill, specify the point where the charge breaks away from the periphery of the mill. This is called the "angle of break", which is measured up the mill periphery from the horizontal [9]. It is assigned that the best position for falling dawn of the mill charge on its inner lining is at the angle of 5:00 on the watch [119], in order to realize the best breakage of the charge and reduce the centrifugal tendency at higher speeds (the critical speed).

Transient Operations

Stockpiling and blending are two transient, subsequent and engaged ROM handling methods, or they could be considered as one method of two dimensions, to some extent. This handling method for the mined materials could be applied for many utilities such as:

- Overcoming the heterogeneity of the different ROM physical properties, which is considered the most important application.
- Stabilize ore feeding to certain operations, especially within processing, such as primary crushing, milling or the subsequent concentration steps.
- Compensate for ROM shortage due unsteady mining rates resulting from the scheduled equipments maintenance or the unscheduled mining stoppage.
- Accommodate for redundant ROM due to processing facilities limitation in capacity or due to any occasional or accidental plant jam or breakdown.

This strategy will guarantee continuous, stable and smooth processing throughputs, in the event of occurrence for one of the previous cases.

Blending is principally used for ore feeding standardization for some sensitive processes. From the important sensitive processes, which require feed standardization, are crushing and milling, in which the most important factors are ore grain size, hardness and grind-ability.

Another sensitive process, which requires feed standardization, is the concentration process (e.g. floatation, ion exchange…), in which the most important factor is the ore grades and mineral contents.

An advanced perspective for stockpiling and blending is accompanied by the selective mining and ROM monitoring and tracking considerations, (Fig. 20), in order to differentiate the mine extraction zones and blocks according to their physical properties, for further processing adjustments.

Concerning blending, piles are typically constructed with many thin layers and should be reclaimed with specialized equipment that cuts across as many layers as possible to maximize the blending efficiency, in the case of necessity to attain a certain property magnitude. This is such as stabilizing the ore content before further accurate mineral concentration process.

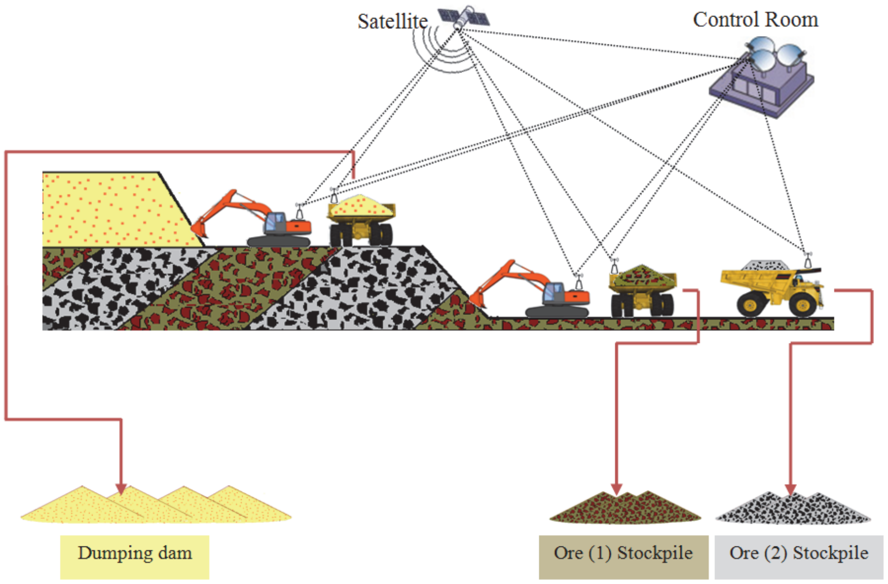


Fig. 20: Online selective mining, hauling and stockpiling.

Blending strategy, according to the previously stated perception, can be illustrated in Figure 21, which demonstrates an ore characterized by three different-range categories (A, B, C) for a given certain physical property such as liberation grain size, hardness, work index, etc. In this figure, for example, ten blending strategies, which are summarized in Table 5, are illustrated and ranged from pure A to pure B to pure C.

Table 5: Blending strategy.

Blending Code	Ore Type (Property) (%)		
	A	B	C
1	100	0	0
2	0	100	0
3	0	0	100
4	0	66	33
5	33	66	0
6	0	33	66
7	66	33	0
8	33	0	66
9	66	0	33
10	33	33	33

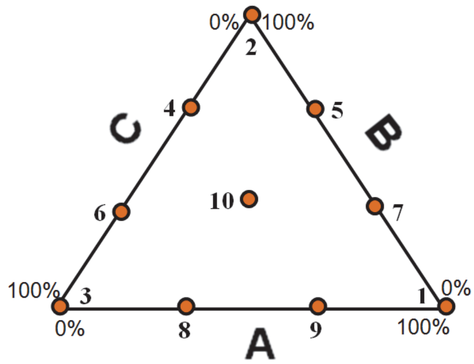


Fig. 21: Coding triangle for blending design.

Special considerations within stockpiling and blending:

➢ Once the ore is excavated and stockpiled, it usually looses its spatial context and much of the information about the site history. In order to prevent this loss, the most efficient approach is to use in situ samples, along with qualitative information about the site history and usage. This can be done, for example, by photo-analysis technology engaged by GPS monitoring and tracing.

➢ To minimize the possibility of misclassification of ROM, the size of stockpiles should be kept relatively small, especially when the scheduled mining area shows heterogeneity in the considered ore property. The actual final mean of the considered ROM property represented within a certain stockpile, should be in the acceptable range of the 95% statistical confidence, regarding the real (actual in-situ) value which is mapped in the ore 3-dimentional model.

3.3.3 Mining and processing special indicators

Cut-Off Grade and Net Present Value Indicator

Cut-off grade (COG) is defined as the grade, which separates the mineralized rocks from barren rocks and is considered the lowest grade of ore in a deposit that will recover all the mining and processing costs. The resource potential will therefore be determined by this grade.

The resource tonnage is calculated from the ore grade distribution and plotted on a grade–tonnage graph, where the material above COG will be used to develop the mine plan and the blocks are scheduled for extraction. The schedule will be affected by location and distribution of ore in respect to topography, elevation, mineral types, physical characteristics, grade-tonnage distribution, and direct operating expenses associated with mining, processing, and concentration.

Figure 22 shows how errors in predicting properties of the resource, which may result in a considerable misclassification of it [128].

The ellipse represents a band of confidence (95%) between the estimated and actual values. It is apparent that there are two areas where material is classified correctly and two areas of misclassification. Misclassified waste has a grade below the COG but it is overestimated to be above it. This misclassified waste will be mined and processed as ore. Likewise, misclassified ore is classified as waste and dumped without extracting any metal, with the subsequent loss of revenue. Thus it is so important to making as accurate an estimate as possible for the COG.

Within estimation of the COG, the objective function is to maximize the incremental present value which can be represented mathematically as follows [107]:

$$NPV_{Max} \sum_{i=1}^{n_y} \frac{v_i}{(1+r)^i} \tag{2}$$

Where: i year indicator of the mine life n_y, v incremental present value (\$/y) and r discount rate.

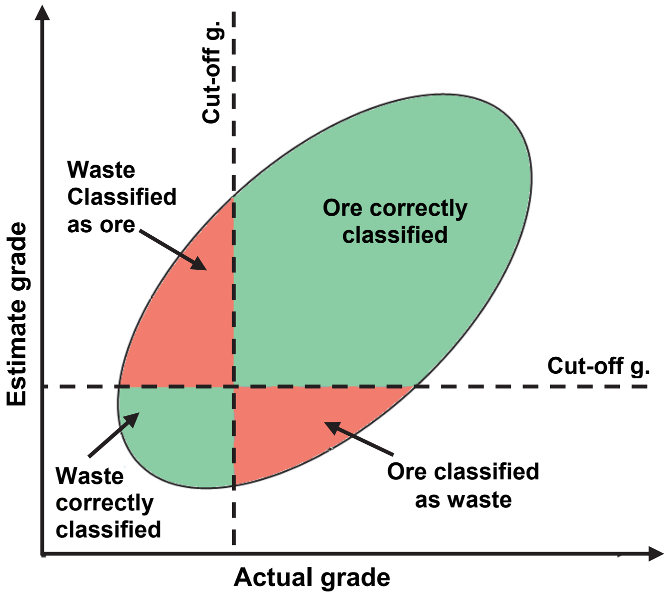


Fig. 22: Confidence ellipse for the grade value estimation [128].

Drilling specific energy indicator

Drilling parameters can be used to estimate rock drill-ability or blast-ability. The most common approach, to predicting this, is the concept of specific energy, which is the work done per unit volume of rock drilled.

Considering the following drilling parameters [122]: penetration rate (*PR*); drilling torque (*DT*); revolution number (*N*); cross section area of drill hole (A_{hole}); and pull-down force (*F*), the work done per minute is given by:

$$W = F(PR) + 2\pi N(DT) \tag{3}$$

The volume of material excavated rate is given by

$$V = A_{hole} \times (PR) \tag{4}$$

Then, the drilling specific energy SE_d is given by:

$$SE_d = \frac{w}{v} = \frac{F \times (PR) + 2\pi N \times (DT)}{A_{hole} \times (PR)} = \frac{F}{A_{hole}} + \frac{2\pi N \times (DT)}{A_{hole} \times (PR)} \tag{5}$$

Specific energy can be thought of as having two components, one due to the pull-down force and another due to the torque. The contribution from the pull-down force is very small compared to that from the torque (< 5%) [64, 112, 122]. Then, neglecting the first term will lead to the following equation:

$$SE_d = \frac{2\pi N \times (DT)}{A_{hole} \times (PR)} \tag{6}$$

Considerable variations of the specific drilling energy with depth can be inferred from Figure 23, which shows the variation of the drilling advance through different rock formations along the drilling profile [39].

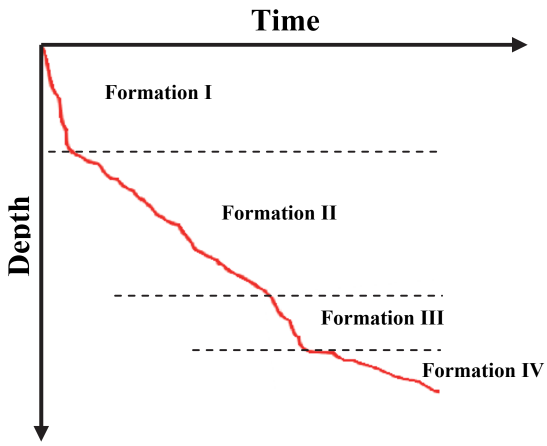


Fig. 23: Drilling advance (time) along the drilling profile (depth) [39].

From the Figure, it can be seen that the profile inclinations, through the different layers, tend to be reduced as the depth increases. This indicates that the specific energy, in general, has a gentle upward trend with depth, which may be due to increasing of the rock hardness due to the confining pressure. Sudden changes, at specific depths due to lithologic changes or fractures, can be shown also. Thus, nature and properties of the discontinuities based on drill monitoring data could be identified.

For the purpose of correlating specific energy with blast fragmentation, a single specific energy value can be obtained for each hole by averaging all the interval specific energies of each hole.

3.4 Introduction to the dynamic modeling and simulation softwares

For the purpose of investigation and achievement of the integrated mining and processing optimization, modeling of the problems and the perceptions is the best way to highlight the weak and critical locations and trying to introduce the best suitable solutions.

A simulation model is the refinement and closure of a set of dynamic hypotheses to an explicit set of mathematical relationships. Simulation models generate behavior through simulation; and it provides a laboratory, in which understanding of how different elements of structure determine behavior could be experimented.

Belonging to the dynamic modeling software, some softwares are reviewed in order to choose the most suitable one to be used in this study as will be stated in the followings.

- (*iThink*), from *STELL*A (Systems Thinking for Education and Research) [142], is a dynamic modeling software, in which Stock and Flow diagrams support the common language of the systems and provide insight into how systems work. Stock types enable discrete and continuous processes with support for queues, enhanced conveyors, etc. Causal Loop Diagrams present overall causal relationships and the model equations are automatically generated and made accessible beneath the model layer. Built-in functions facilitate mathematical, statistical, and logical operations. The software supports hierarchical model structures that can serve as building blocks for the model construction.
- *SolveIT* Software's *APS and SCNO* applications for (resource-to-ship) simulation [143] allows users to change assumptions, parameters, business rules, and/or constraints to generate multiple scenarios. Assets (e.g. mine stockpiles, process plant modules, rail infrastructure, and port components such as stockyard rows, ship loaders etc.) can be added to run multi-time horizon

simulations. The simulation capability of *APS and SCNO* applications can also be augmented by non-linear optimizer, which allows for optimization of the simulation model parameters.

- *Vensim-PLE* for Windows (Version 5.11A), Copyright© 1988-2010, Ventana Systems Inc. [127], is chosen to be used in this study for the following reasons:
  - *Vensim-PLE* is a dynamic modeling and simulation software, which provides a flexible way of building system dynamic and simulation models from Causal Loop or Stock-and-Flow diagrams.
  - Within *Vensim-PLE*, when a model is built, its behavior could be thoroughly explored, dynamically, hence it realizes a self-validation.
  - Advanced features of Vensim, such as sensitivity testing and model optimization are also provided to experience the various outputs.
  - In addition to building models, Vensim can perform simulation tasks by changing to simulation setup and SyntheSim mode from the main build window.
  - *Vensim-PLE* has a free version, which is used in this study, which is considered a further advantage for its choice.

3.5 Particular concepts belonging to the chosen modeling software

By connecting Words with Arrows, relationships among system variables are entered and recorded as causal connections. This information is then used by an *Equation Editor* to help in formation of a complete Dynamic model. The built model can be analyzed throughout the building process, looking at the Causes and Uses of a variable, and also at the Loops involving the variable. In the followings, the particular concepts belonging to *Vensim-PLE* Software are presented.

Events, behavior and structure

Mining and processing are filled with events: drilling achievement, blasting performance, loading and hauling efficiency, stocking capacity, crushing and grinding reliability, and so on. Because of their prevalence, events tend to fill the discussions.

One step back from events is the idea of behavior patterns. A behavior pattern is something that connects together a long series of events over time. Stepping away from events and begin considering patterns of behavior, questions such as "what caused ..." are given a different and much deeper meaning. We are looking for sources of pressure and imbalance that cause things to change.

Structure is the set of physical and information interconnections that generate behavior. For example, inventory is the accumulation of production less shipments. Workforce changes with hires and attrition; and hiring are based on the targeting of production to meet demand and correct inventory imbalances. The result of this is that the inventory level moves up and down (behavior) and when inventory is extremely so much the relative profit decreases, then the production should be diminished (an event). Structure determines behavior; and events are snapshots of that behavior.

The event – behavior – structure distinction is an important tool for understanding and working with problems. Ultimately, successful policies and interventions need to be changed to structure, so that behavior is improved and bad events become less frequent. System dynamics and Vensim provide tools to represent structure, and understand how it determines behavior.

System dynamics process progression

There are some basic practices that are commonly used for developing good quality system dynamics models. The following steps, illustrated in Figure 24, form a guideline for the process progression:

Issue statement and variable identification

The issue statement is simply a statement of the problem that makes it clear what the purpose of the model will be. Clarity of the purpose is essential for development of an effective model.

Variable identification of some key quantities should be done in order to be included in the model for being able to address the issues at hand. It could be useful just to write down all of the variables that might be important and then ranking them in mind, in order to identify the most important ones.

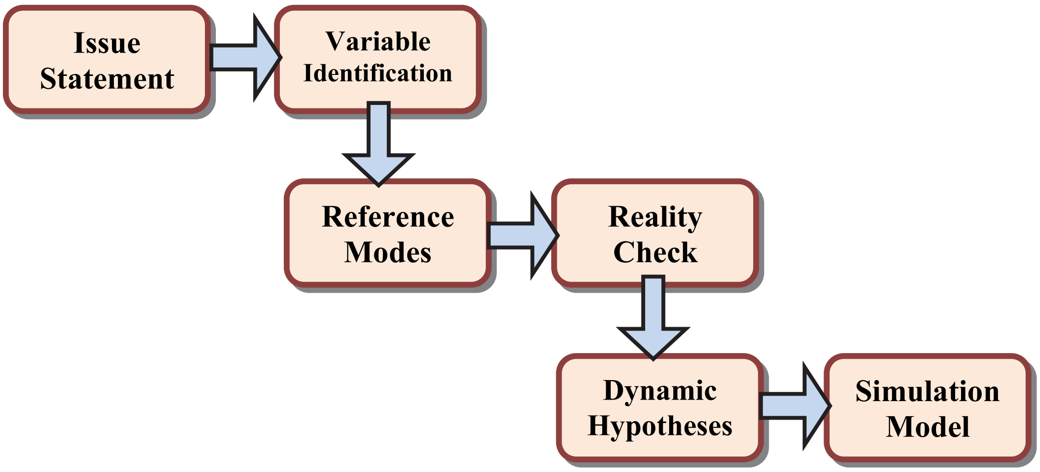


Fig. 24: Progression steps for system dynamics models development.

Reference modes and reality check

A reference mode is a pattern of behavior over time; and it is drawn as graphs over time for key variables. Reference modes show a particular characteristic of behavior that is interesting and can refer to past behavior or future behavior. They should be drawn with an explicitly labeled time axis to help refine, clarify and bound the problem statement.

Reality Check is a definition for some statements about how things must interrelate. These include a basic understanding of what actors are involved and how they interact, along with the consequences for some variables of significant changes towards other variables. Reality Check information is simply recorded as notes (mental notes) about what connections need to exist.

Dynamic hypotheses and simulation model

A dynamic hypothesis is a theory about what structure exists that generates the reference modes. A dynamic hypothesis can be stated as a Causal Loop Diagram, or as a Stock-and-Flow Diagram; and it can be used to determine what will be kept in models, and what will be excluded.

In Simulation Setup mode, all model constants and Lookups will be highlighted and could be temporary changed to the values to be used for a simulation. Vensim works using a “workbench” metaphor. At all times there is a “Workbench Variable”, which is the model variable that some tools automatically apply to it [127].

3.6 Main tools, components and constituents of the used software

Vensim-PLE is consisted of various and many tools. As any Windows supported software, Vensim contains many user assistants such as: the Build Window, Sketch Tools and Title, Status, Menu and Tool Bars; in addition to Analysis Tools and Simulating Bars.

In the following paragraphs, some of the main components and definitions will be graphically and exemplary illustrated.

Causal-Loop diagramming

Causal loop (or influence) diagrams are called that because each link has a causal interpretation. An arrow going from A to B indicates that A causes B. Causal loop diagrams can be very helpful in conceptualizing and communicating structures. They do not show accumulations (levels or stocks) in a system. Both Causal loop diagrams and stock-and-flow diagrams are not simulation models, but the simulation models is formed after attaching algebraic relationships to all the variables appearing in these diagrams.

Figure 25 illustrates a simple positive causal feedback loop, of a length 1, for the variable level A, which consists of a main integration equation (included in the equation editor), one constant (which could be here a lookup table or graph) and other factors, which if included in the model will produce more other loops.

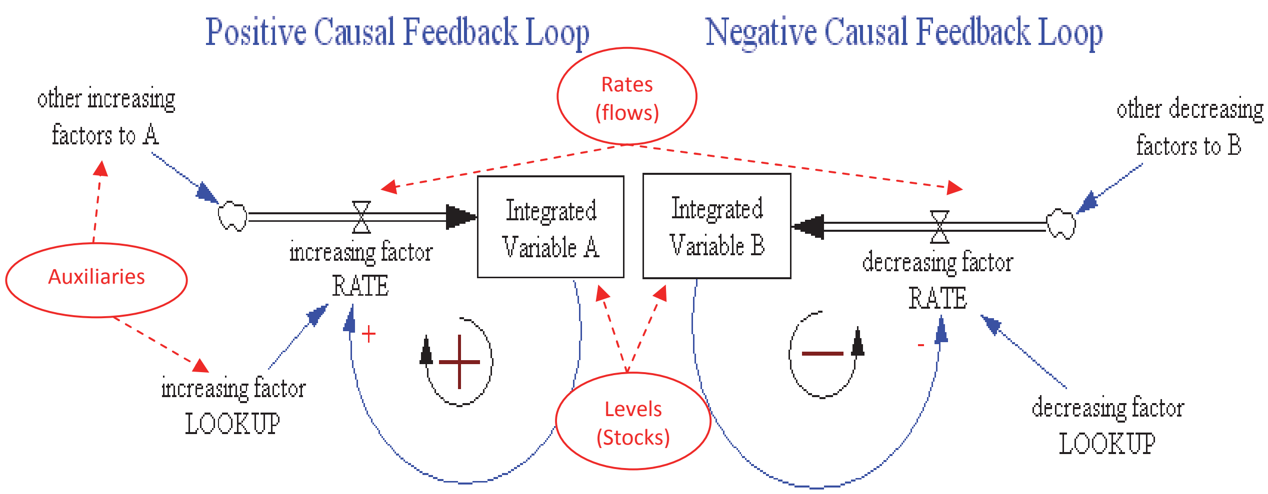


Fig. 25: Simple positive and negative causal feedback loops.

The same description can be assigned to the other simple negative causal feedback loop, in the same figure.

Causal tracing with trees

Causal Tracing is a powerful tool for moving through a model tracing what causes something to change. The simple causal tree for the above positive loop is illustrated in Figure 26, which help in tracing the other interconnected affecting factors or constants.

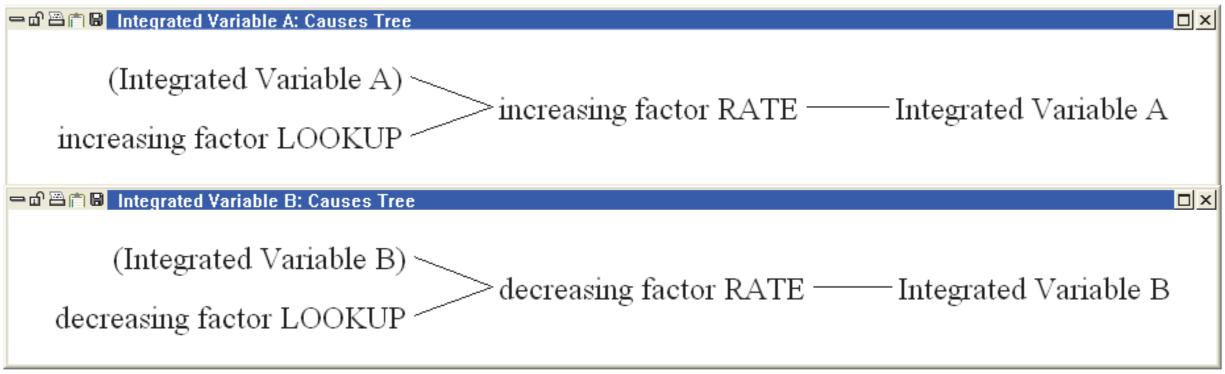


Fig. 26: Simple causal tree for the illustrated positive and negative loops.

The causal tree for the other negative loop is illustrated also in the same figure. Within complicated models, which contain many interfered and overlapped loops, the causal tree plays an important role in tracing the hotspots and the problematic points, which may cause some abnormal behavior for certain events.

Stock and Flow diagrams

Stock and flow (or Level and Rate) diagrams are ways of representing the structure of a system with detailed information. Stocks (Levels) are fundamental to generating behavior in a system; flows (Rates) cause stocks to change. Stock and flow diagrams are common steps in building a simulation model, as they help define types of variables that are important in causing behavior.

Levels, Auxiliaries and Rates (Flows)

Levels are also known as stocks or accumulations, which change their values by accumulating or integrating rates. The values of Levels change continuously over time even when the rates are changing discontinuously.

Rates (flows) change the value of levels. The value of a rate is not dependent on previous values of that rate; instead the levels in a system, along with exogenous influences, determine the values of rates. Intermediate concepts or calculations are known as auxiliaries and, like rates, can change immediately in response to any exogenous influences.

The Rate has a single arrowhead, indicating the direction that material can flow (the Rate can only increase the Level). This is only in the diagram, but in the simulation model, the equation governs the direction that material can flow. However, we can use the diagram to indicate whether the flow is intended to be one way or two ways.

Simulation for the model

The modeling process starts with sketching a model, then writing equations and specifying numerical quantities. Next, the model is simulated with simulation output automatically saved as a dataset. Finally, the simulation data can be examined with Analysis tools to discover the dynamic behavior of variables in the model.

Normal model construction follows a pattern of create, examine, and recreate, iterating until the model meets the requirements. Debugging (making a model simulate properly) and model analysis (investigating output behavior) both play a part in refining the model.

The behavior of a simulation model in Vensim is just determined by the equations that govern the relationships between different variables. The diagram of a model (causal loop or stock-and-flow) is a picture of the relationships between variables, and then Vensim enforces consistency of the diagram and the model equations.

3.7 Assumed case study for the model construction

As working with a dynamic modeling program, such as Vensim-PLE, requires experimenting real numerical parameters, in order to be inputs, engaged with the suitable and reliable functions and equations, a case study data is utilized. This data are principally used in building up the main Loops, Stock and Flow diagrams, with their Levels, Auxiliaries and Rates, which are explained in the upper sections. In the same time, the utilization of this real case study data is considered a validation for the reliability and the soundness of the model, which could be judged from the ease of generating results (outputs) and the pragmatic interpretations of them.

A case study data, which are generated from a real copper open pit mine with some other more assumptions for the missed information, are considered here in this thesis.

Reasons for a (metalliferous-ore deposit) choice as a case study

A copper-porphyry deposit is selected to be a case study data for using in the modeling in this study and the reasons for this choice are as follows:

- The metal-minerals mining is considered the most important and widely distributed due to their principal participation in the modern civilization and industry;
- Overall increasing in the market demand and price for the metal commodities;
- The huge investments and expenses in the metal-minerals mining and processing;
- The special need for fine ground products during metal-minerals processing, for efficient liberation to the valuable metal portions, especially with the case of copper minerals processing;
- Usually, the metal-minerals deposits contain multi-metal ores or different ore types for the same metal, especially with the case of the copper-porphyry deposits, which enriching thoughts for the integrated mining and processing optimization;

It should be mentioned that just the available applicable data and parameter values will be considered and all other missed operational parameters and the technological criteria are comparatively assumed with different configurations, in order to trace the different factors variation effects on the yields of the model.

It should be also mentioned that these data assumptions is intended to be not definitely affecting the model reliability, that the model is intended to be used with any other input mining data, or technological concepts to experience the response in the final results.

4. Calculation basics for Applying Dynamic Modeling and Simulation for the Mining and Processing Operations

4.1 The modeling construction strategy

Constructing, examining, and modifying the model-parts is following an iterative approach, starting from simple models with few feedback loops and particular details, which gradually allow the construction of the main working simulation model. Model-parts or, as called in this thesis, the sub-models will be divided into three main divisions:

- Drilling and Blasting;
- Loading and Hauling; and
- Crushing and Grinding.

These intended divisions are the same as classified before (CH. 3), in order to highlight the most effective natural and operational parameters in each operations group for mining and processing. Another reason for this particular division is to illustrate the sensitivity and feedbacks due to any manipulating in certain operations parameters and examine their effects on the same and the other operations.

The working model can then be modified and improved as necessary to show the desired level of details and complexity.

Normal model construction follows a pattern of create, examine, and recreate, iterating until the model meets the requirements. Debugging (making a model simulate properly) and model analysis (investigating output behavior) both play a part in refining the model.

During simulation, dynamic behavior is stored for all variables in the model. The desired variable of interest can be selected and more detailed data about it can be displayed by the appropriate analysis tool in order to trace the effect of its respond to changing with any other linked variables.

4.2 Construction of the [*Reference-Mode*] model

4.2.1 Dynamic modeling and simulation for the drilling and blasting operation

On designing or analyzing a model or a simulation, it is useful, at first, to represent the system data graphically. Block Diagrams are a useful and simple visual method for accomplish this important step.

The block diagrams are ways of representing relationships between the different components in a system, because these schematic diagrams can capture and visualize input-output relations and help in understanding flow of the essential information needed to implement the required model.

Considering that every sub-model can be described as a system, these systems can be then combined into a single representative individual system, with transfer dependent and independent information.

Block diagram and data preparation for the inputs and outputs parameters

The block diagram for the sub-model (*Drilling and blasting*) is shown in Figure 27. In general, the independent data will be the output of each sub-model (system), which could be considered as a final output concerning this system. While, the dependent data are the intermediate data which are outputs of one or more system and are in the same time also inputs to other systems.

It should be mentioned, especially within the 2nd and the 3rd sub-model, that the intermediate and inter-related data are much more than the final outputs for each one.

From another point of view, the block diagram in itself gives good information of the final structure of the model, e.g. how sub-systems are connected.

The uppermost modules are always the sources, from which the input parameters are generated, while the lowermost modules represent the two output types (results), the dependent and the independent. In this sub-model, the input sources are the available operational, technical, and economical data, in addition to those, which are generated from the ore-deposit model and the planning data.

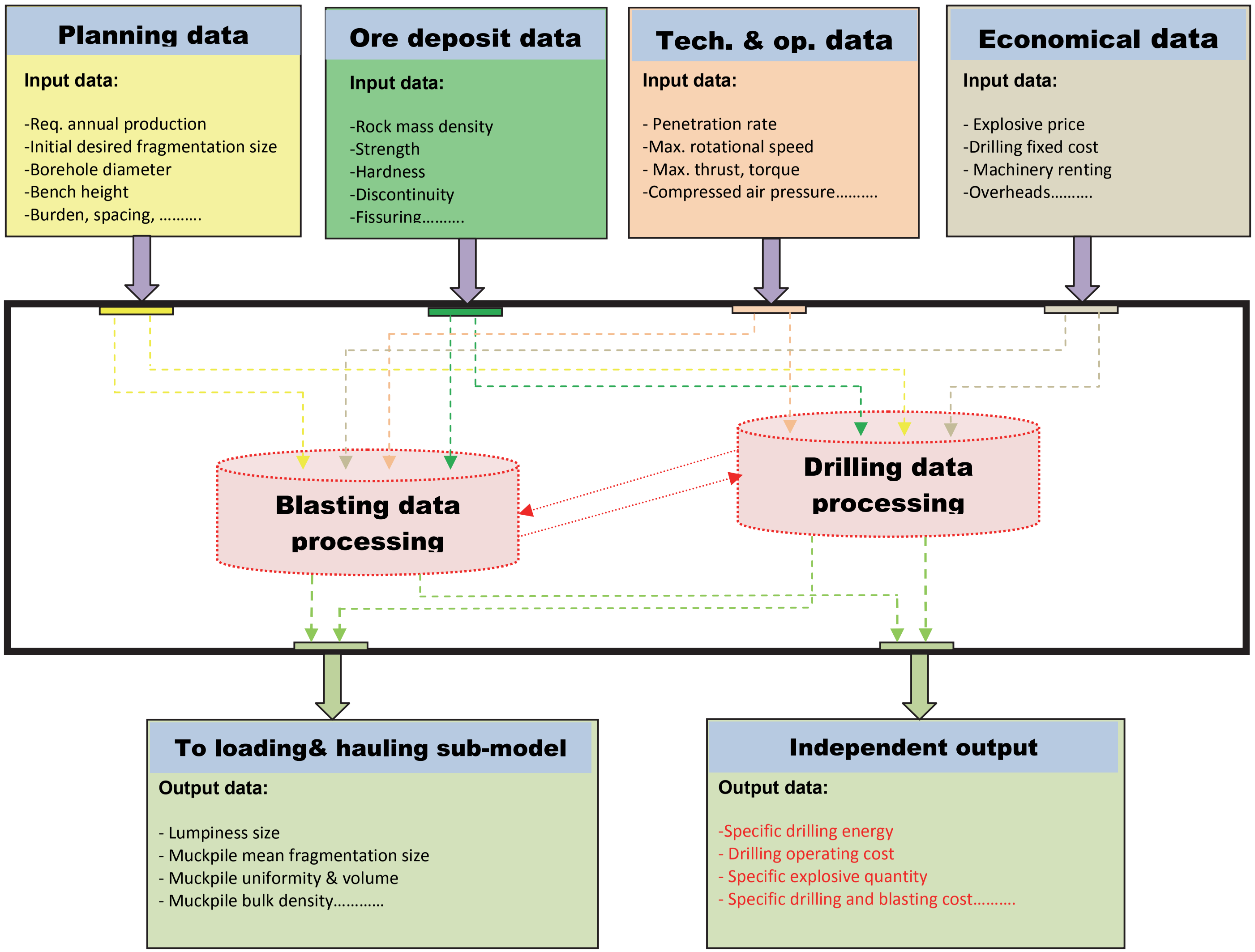


Fig. 27: Block diagram for the data processing within *Drilling and Blasting* sub-model.

The intermediate large block shows how and to where the data are directed, in order to be processed within the model. It also includes the much more other intermediate and calculated items.

Processing of the data will be detailed in the next section, which includes the mathematical formulas, lookups and algebraic equations needed for the model construction.

The *Drilling and Blasting* sub-model construction

The main goal from this sub-model is to produce a link between the rock properties and natural parameters from one side and the measurable drilling technical data and the blasting parameters for the other side. The real image will be reflected by the economic results of the blasting operation in addition to the other intermediate results, such as mean fragmentation size, muck-pile uniformity, swelling….etc.

These mathematical link all the inputs and the generated intermediate parameters into one net of information, in order to generate finally the output results.

The Kuz-Ram model is used here [70], in order to calculate the rock mean fragmentation size for a certain applied explosive energy. It was developed by Cunningham [29, 30], who modified Kuznetsov's equation for ANFO based explosives to estimate the average fragment size X_{50}, (cm), as:

$$X_{50} = K_{rock} \times \left(\frac{V_0}{M_{sp.ex}}\right)^{0.8} \times M_{ex}^{\;0.167} \times \left(\frac{115}{RWS}\right)^{0.633} \tag{7}$$

where: RWS relative weight strength of explosive, ANFO=100 and TNT=115

M_{ex} quantity of explosive in one blast hole, (kg); V_o bank volume, (m^3);

$M_{sp.ex}$ powder factor, (kg/m^3); and

K_{rock} rock factor (blastability index) [16,41]: which is equal to 7 for medium rocks, 10 for hard, highly fissured rocks, and 13 for the very hard, weakly fissured rocks. It can be also calculated as:

$$K_{rock} = 0.06 \times (RMD + K_{joint} + RDI + K_{hard}) \tag{8}$$

Where: *RMD* rock mass description, K_{joint} is the joint factor, *RDI* rock density index, and

K_{hard} is the hardness factor.

These factors are calculated from the geological data and the mechanical characteristics of the blasted rocks [54]. Here in the model, the blastibility is estimated from the lookups, which relate the rock geological characteristics such as strength σ, discontinuity (spacing) ε, bank density ρ, etc., with the mining rock mass rating (RMR), which is explained before, (Appendix 1).

The rock mass strength effect and also the fissuring or the discontinuity effects of the drilled and blasted rocks are represented as lookups within the model. The lookups, as explained in the previous chapter, are the introduced tables or special curves of previously known points, to the model.

Kuznetsov's equation is combined with the Rosin-Rammler equation [30] to predict the entire size distribution (the lumpiness size is assumed to be the X_{80} size, for example) as follows:

$$R = 100 - e^{-0.693\left(X/X_{50}\right)n} \tag{9}$$

Where: *R* percentage smaller than *X,*

X size of rock, *n* uniformity exponent [47] and it can be estimated as follows:

$$n = \left(2.2 - \frac{14\times B}{D}\right) \times \left[\frac{1+\frac{S}{B}}{2}\right]^{0.5} \times \left(1 - \frac{w}{B}\right) \times \left(\frac{L}{H}\right) \tag{10}$$

Where: *B* blasting burden (m), *S* blast hole spacing (m), *D* blast borehole diameter (mm),

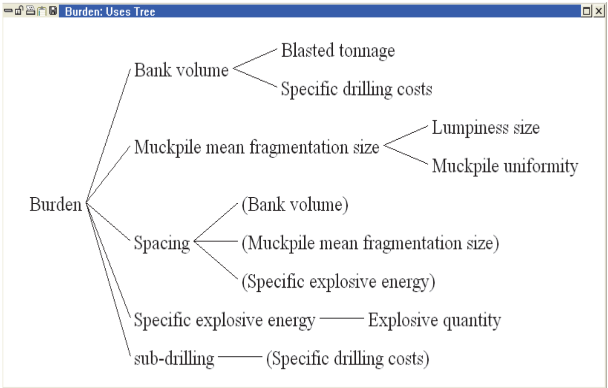
w standard deviation of drilling accuracy (m), *L* total charge length (m), and *H* bench height (m).

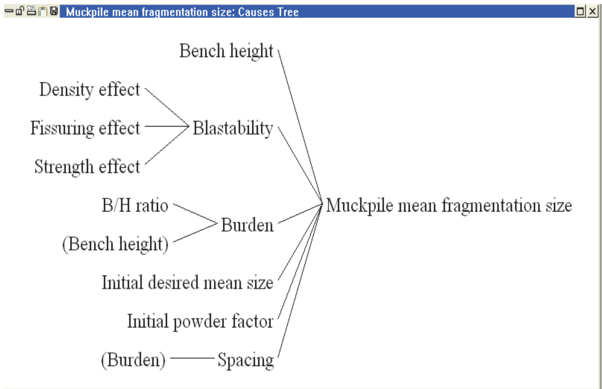
The uniformity coefficient usually varies between 0.8 and 1.5 and, from the previous equation, it has a directly proportion with the charge length, hence:

$$n \propto \frac{1}{X} \tag{11}$$

Therefore, a lookup is installed within the model to represent this relationship between the fragmentation size and the fragments uniformity coefficient.

It should be mentioned that, from the tools, which is essential in helping to solve most encountered problems during the construction of the model, are the (uses) and the (causes) tree. Figure 28 shows an example for a uses tree and a causes tree for two different parameters.





(a) (b)

Fig.28: A screenshot illustrates a (uses) and a (causes) tree for two related parameters.
(a) The uses tree for the Burden parameter; (b) The causes tree for the Muckpile mean fragmentation size.

As explained in the previous chapter, these two opposite tools make a good tracing for the different inter-related parameters, in order to trace any weakness or problems belong to functions circulation, modeling errors, units unbalance or any other warnings.

The inverse of Kuznetsov's equation is used to estimate the required explosive energy which should be used in order to produce a certain pre-planed mean fragmentation size.

Given the bench height H, (m) and the blast borehole diameter D, (m), the other main blasting parameters can be estimated from the followings [35, 36, 41, 57]:

$$\frac{B}{H} = 0.3 \qquad (12)$$

$$S = 1.25 \times B \qquad (13)$$

$$\Delta = 0.2 \times B \qquad (14)$$

$$V_o = H \times B \times S \tag{15}$$

Where: B burden, (m); S spacing, (m); and Δ sub-drilling, (m).

Then is the quantity of explosive in one blast hole M_{ex}, (kg) and the blasted tonnage due to one bore hole M_o, (ton), are estimated from the following:

$$M_{ex} = M_{sp.ex} \times V_o \tag{16}$$

$$M_o = V_o \times \rho \tag{17}$$

Where: $M_{sp.ex}$ powder factor, (kg/m^3); and ρ bank rock density, (t/m^3).

As explained in the previous chapter, after laying out a certain group of related parameters and link them by the arrows, the appropriate units and equations are introduced for the real linking of them together functionally. Figure 29 shows an example for an equation introduced to one parameter within the model.

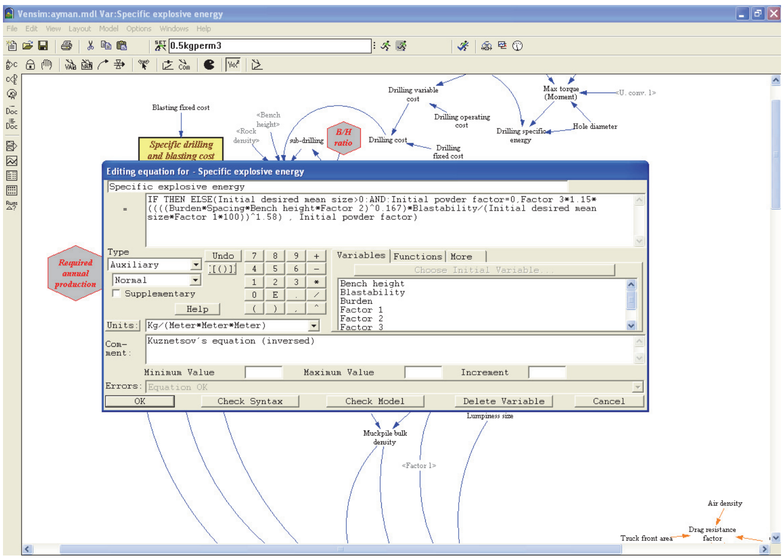


Fig.29: A screenshot for an equation introducing example to the model.

Afterwards, the specific blasting cost SC_b, (\$/t) can be is calculated as:

$$SC_b = \frac{(C_{b.fix} + M_{ex} \times \$_{ex})}{M_o} \tag{18}$$

Where: $\$_{ex}$ explosive price, (\$/kg); and $C_{b.fix}$ blasting fixed cost, (\$/bore hole).

The blasting fixed cost is such as detonator, poster, initiation, overheads, others, etc., for each hole.

The blasting holes number $n_{b.h}$ is calculated as:

$$n_{b.h} = \frac{M_{b.ore} \times (S_r + 1)}{M_o} \tag{19}$$

Where: S_r stripping ratio, (t/t), (Dmnl); and $M_{b.ore}$ ore produced per blast, (t), which can be calculated as:

$$M_{b.ore} = \frac{M_{an.ore}}{n_{blast} \times n_d} \tag{20}$$

Where: n_d annual working days; n_{blast} blasting frequency; and

$M_{an.ore}$ required ore annual production, (t).

The most common approach to predicting drillability and blastability is the drilling specific energy SE_d (t.m/m^3), which can be calculated from the following [1, 65]:

$$SE_d = \frac{8 \times (DT)}{D^2 \times (PR)} \tag{21}$$

Where: PR penetration rate, (m/h); and

DT drilling torque, (t.m/h), which can be calculated as:

$$DT = 60 \times \pi \times D \times N \times F_{max} \tag{22}$$

Where: F_{max} drilling-rig maximum thrust, (t); and N revolution number, (rpm).

The drilling cost C_d ($/m), can be calculated by the followings [7]:

$$C_d = C_{d.var} + C_{d.fix} \qquad (23)$$

$$C_{d.var} = \frac{C_{d.op}}{PR} \qquad (24)$$

Where: $C_{d.fix}$ drilling fixed cost, ($/m); $C_{d.op}$ drilling operating cost, ($/h); and $C_{d.var}$ drilling variable cost ($/m).

Then, the specific drilling cost SC_d ($/t) can be calculated from the equation:

$$SC_d = \frac{C_d \times (H+\Delta)}{M_o} \qquad (25)$$

Finally, the specific drilling and blasting cost $SC_{d\&b}$ ($/t) can be calculated as:

$$SC_{d\&b} = SC_d + SC_b \qquad (26)$$

A screenshot for the *Drilling and Blasting* sub-model is illustrated in Figure 30. The arrows, which are connecting between the parameters, indicate that there are equations relating between them. The hexagonals include the main input parameters, which can be manipulated, while the main independent outputs are included in the yellow rectangles. Also, the arrows, which go down connect certain intermediate parameters to the loading and hauling section.

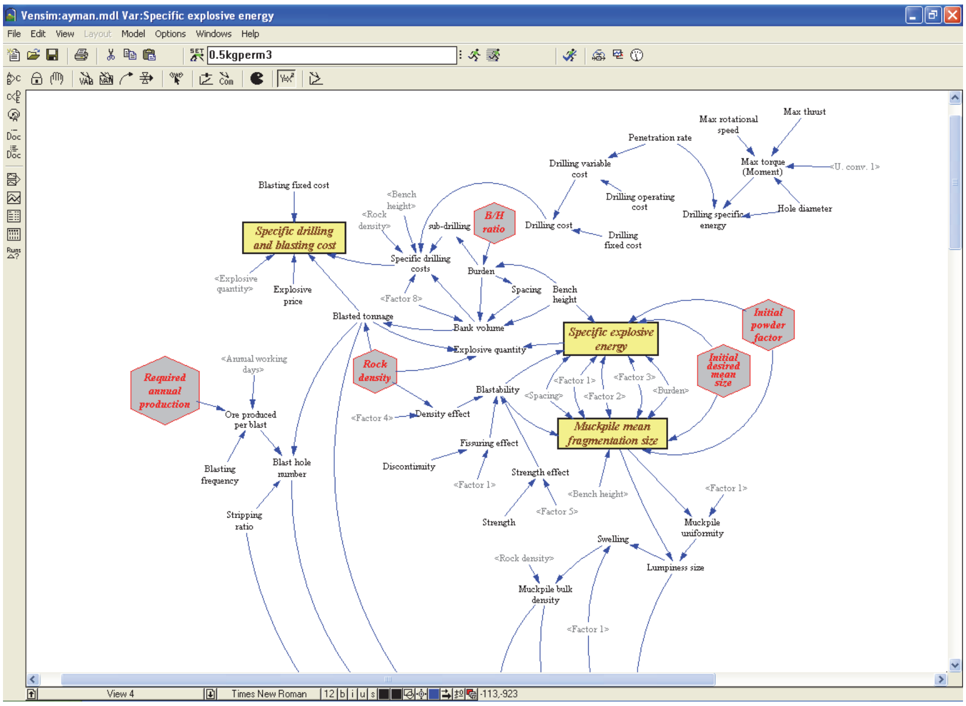


Fig. 30: Screenshots for the *Drilling and Blasting* sub-model.

4.2.2 Dynamic modeling and simulation for the loading and hauling operations

Block diagram and data preparation for the inputs and outputs parameters

The block diagram for the sub-model (*Loading and Haulting*) is shown in Figure 31. In general, the independent data will be the output of each sub-model (system), which could be considered as a final output concerning this system. While, the dependent data are the intermediate data which are outputs of one or more system and are in the same time also inputs to other systems.

The block diagram gives good information of the final structure of the model and provides a view of how the sub-systems are connected. As it is discussed before, the intermediate and inter-related data are much more than the final outputs of this sub-model.

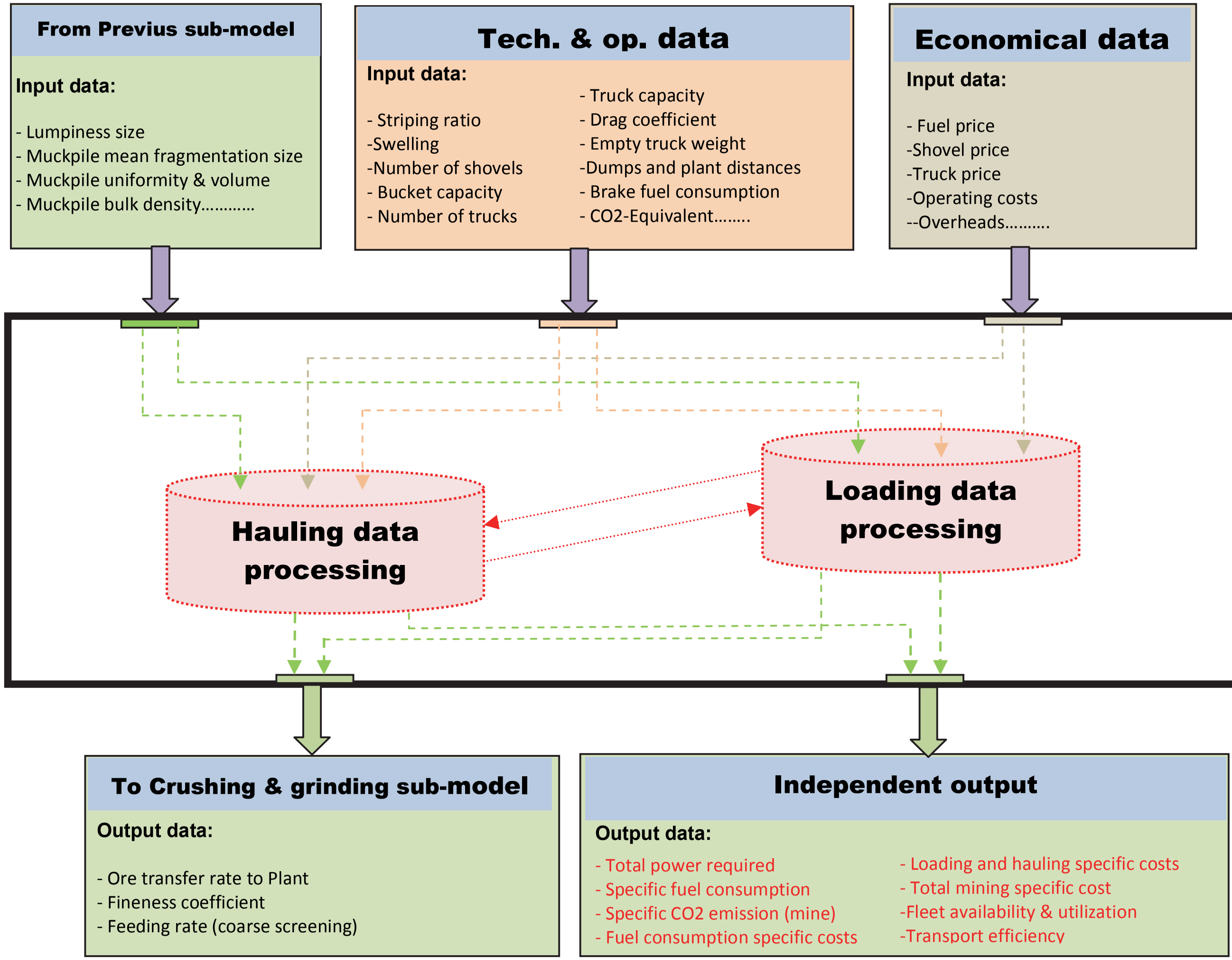


Fig. 31: Block diagram for the data processing within *Loading and Hauling* sub-model.

The uppermost modules indicate the sources of the input parameters which are generated from the previous sub-model, in addition to the available operational, technical, and economical data, while the lowermost modules represent the two output types (results), the dependent and the independent results.

The intermediate large block shows how and to where the data are directed, in order to be processed within the model. It also includes the much more other intermediate and calculated items.

Processing of the data will be detailed in the next section, which includes the mathematical formulas, lookups and algebraic equations needed for the model construction.

The *Loading and Hauling* sub-model construction

The main goal from this sub-model is to produce a link between the fragmentation size of the blasted rocks, the rock properties and natural parameters from one side and the delivery rate of ROM to the plant according to its maximum capacity and the planed production requirements form the other side.

The real image will be reflected by certain economic and environmental results such as the fuel consumption, the fleet utilization, and the greenhouse emissions, in addition to the other intermediate results, such as pay load, shovel loading rate, bucket filling factor, total cycle time….etc.

The following mathematical formulas link all the inputs and the generated intermediate parameters into one net of information, in order to generate finally the output results.

Firstly, the required excavation rate V_{excav}, (m^3/h), can be calculated as:

$$\dot{V}_{excav} = \frac{M_{an.ore} \times (S_r + 1) \times (1 + \omega)}{n_d \times n_h} \tag{27}$$

where: n_h planed daily working hours, (h); and ω swelling factor, (Table 6).

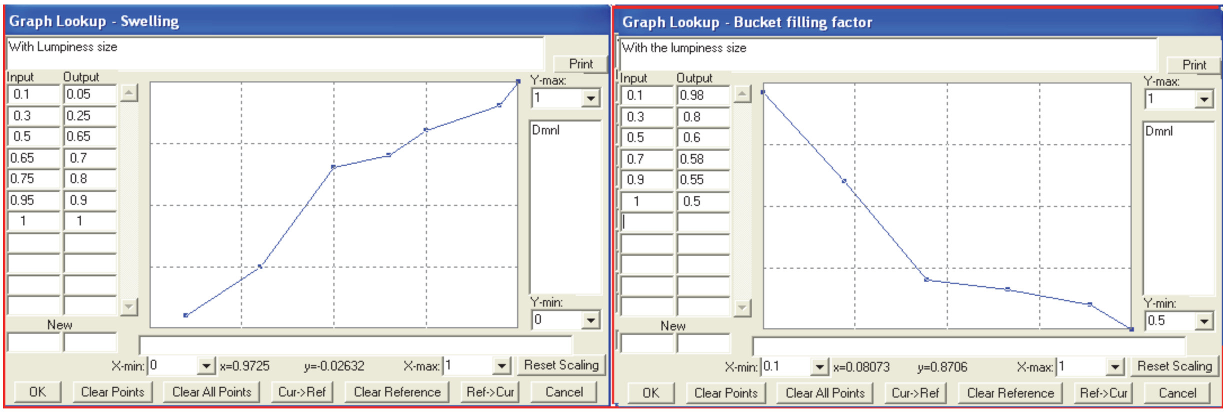
The swelling factor is related to the lumpiness size (X_{80}) of the muck-pile according to Table 6, so a lookup is made within the model in order to represent its value.

Then, for a certain shovel type, the volume excavated rate V_{sh}, (m^3/h), can be obtained from:

$$\dot{V}_{sh} = V_{bucket} \times K_f \times n_{buckets} \times \eta_l \quad (28)$$

where: V_{bucket} bucket capacity, (m^3); η_l loading efficiency; $n_{buckets}$ number of buckets per hour (shovel loading rate); and K_f is the filling factor.

It should be noticed that the shovel loading rate and the bucket filling factor are designed in the model as two lookups which are related to the lumpiness size of the excavated muck-pile, according to a referenced table. Figure 32 shows a lookup example for the bucket filling factor and other for swelling, related to the lumpiness size. The values for the swelling and filling factors are generated from a reference table [80, 135], (Table 6).



(a) (b)

Fig. 32: A screenshot for an example of two lookups within the 2nd sub-model:
(a) The Swelling parameter; (b) The Bucket filling factor.

Therefore, the required number of shovels and the required number of trucks, for a certain truck type, can be estimated as [68, 83]:

$$n_{sh} = \frac{\dot{V}_{excav}}{\dot{V}_{sh}} \quad (29)$$

$$n_{tr} = \frac{t_{cyc} \times n_{buckets} \times n_{sh} \times V_{bucket} \times K_f}{60 \times V_{tr}} \quad (30)$$

Where: n_{sh} required number of shovels; n_{tr} required number of trucks; V_{tr} truck capacity, (m^3); and t_{cyc} total truck cycle time.

Table 6: The relation between muck-pile swelling and the loading factor [80].

Swell %	Voids %	Filling factor	Swell %	Voids %	Filling factor
5	4.8	0.952	55	35.5	0.645
10	9.1	0.909	60	37.5	0.625
15	13.0	0.870	65	39.4	0.606
20	17.6	0.833	70	41.2	0.588
25	20.0	0.800	75	42.9	0.571
30	23.1	0.769	80	44.4	0.556
35	25.9	0.741	85	45.9	0.541
40	28.6	0.714	90	47.4	.0526
45	31.0	0.690	95	48.7	0.513
50	33.3	0.667	100	50.0	0.500

The total cycle time t_{cyc}, (min), can be estimated as:

$$t_{cyc} = t_l + t_{tr} + t_w \qquad (31)$$

$$t_l = t_{man} + \frac{V_{tr} \times 60}{n_{buckets} \times V_{bucket}} \qquad (32)$$

$$t_{tr} = t_{tl} + t_{te} + t_d \qquad (33)$$

Where: t_l loading time, (min); t_{man} maneuver (scoping) time, (min); t_{tr} travelling time, (min); t_d dumping time, (min); t_{tl} travelling time (loaded), (min); t_{te} travelling time (empty), (min); and t_w waiting time, (min).

The waiting time depends on the shovel loading rate, and hence depends on the muck-pile lumpiness; therefore a lookup is installed within the model in order to represent its relation with the fragmentation size.

The travelling time (loaded and empty) can be calculated from the followings:

$$t_{tl} = \frac{L_c(or{:}L_d)}{v_l} \tag{34}$$

$$t_{te} = \frac{L_c(or{:}L_d)}{v_e} \tag{35}$$

Where: L_c distance to plant, (m); L_d distance to dumping, (m); v_l and v_e loaded and empty travelling velocities, respectively, (m/min).

The loaded and empty travelling velocities can be obtained from the trucks performance charts in their catalogues, according to their loaded and empty weights and the traveling road gradients.

In the model, this routine is rather inversed, thus, at a certain velocity, with gradients assumed to be equal to zero, the different hauled payloads can be traced, in order to determine their effects on the total truck resistance, hence the power consumed and the fuel consumption.

The truck pay load M_{pay}, (t), can be estimated by:

$$M_{pay} = V_{bucket} \times n_{pass} \times K_f \times \rho_b \tag{36}$$

$$\rho_b = \frac{\rho}{(1+\omega)} \tag{37}$$

$$n_{pass} = \frac{n_{buckets} \times t_l}{60} \tag{38}$$

Where: ρ_b muck-pile bulk density, (t/m^3); and n_{pass} number of bucket passes per truck.

The truck pay load has its own constraint that it should be ≤ 110% of its rated payload [21].

The power required by a truck to haul a certain pay load with a certain velocity, here by m/s units, and the other when returns empty, can be calculated from the followings [18, 25, 111, 125]:

$$P_t = P_l + P_e \tag{39}$$

$$P_e = v_e(a \times v_e{}^2 + b \times M_e) \tag{40}$$

$$P_l = v_l(a \times v_l^2 + b \times M_g) \quad (41)$$

$$M_g = M_e + M_{pay} \quad (42)$$

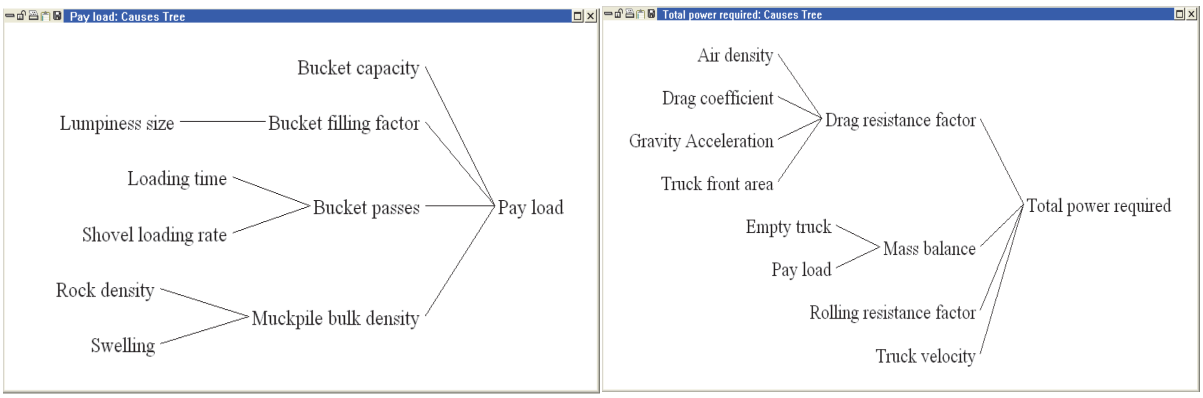
$$a = \frac{1}{2} \times (K_{drag} \times \rho_{air} \times A_f) \quad (43)$$

$$b = g \times K_{roll} \times \cos\theta \pm g \times \sin\theta \quad (44)$$

Where: P_t total power required, (kW); P_l loaded truck power, (kW); P_e empty truck power, (kW); M_g truck gross weight, (t); a drag resistance factor; b rolling and gradient factor, according to mines and quarries roads; ρ_{air} air density, (kg/m^3); θ road gradient, (rad.); g acceleration of gravity, (m/s^2); and the followings are characterized for the truck:

M_e truck empty weight, (t); A_f truck front area, (m^2); K_{drag} drag coefficient; and K_{roll} coefficient of rolling resistance.

An example for a causes and a uses tree for two different parameters is shown in Figure 33. These two tools are important, that they enable the model designer to trace the parameters for a certain group, in the case of any errors or units unbalance encountered during the modeling process.



(a) (b)

Fig. 33: A screenshot illustrates a (uses) and a (causes) tree for two related parameters.
(a) The causes tree for the Pay load parameter; (b) The causes tree for the total power required.

Afterwards, the trip fuel consumption by one truck $M_{f,trip}$, (kg), can be calculated as follows:

$$M_{f,trip} = \frac{BFC}{60}(t_{tl} \times P_l + t_{te} \times P_e) \tag{45}$$

Where: *BFC* brake fuel consumption, (kg/kWh).

The specific fuel consumption SM_{fuel}, (kg/t) will be obtained from:

$$SM_{fuel} = \frac{M_{f,trip} \times n_{trip,tr}}{\dot{M}_{bulk,tr}} \tag{46}$$

$$\dot{M}_{bulk,tr} = M_{pay} \times n_{trip,tr} \tag{47}$$

$$n_{trip,tr} = \frac{60}{t_{cyc}} \tag{48}$$

Where: $n_{trip,tr}$ truck trip frequency, (1/h); and $M_{bulk,tr}$ truck bulk transfer rate, (t/h).

Then, the loading and hauling specific costs $SC_{l\&h}$, (\$/t) can be obtained as:

$$SC_{l\&h} = \left(\frac{\$I_{l\&h} + C_{op.l\&h}}{M_{res} \times (S_r + 1)}\right) + SC_f \tag{49}$$

$$SC_f = f_{\$} \times SFC \tag{50}$$

$$\$I_{l\&h} = n_I \times [n_{tr} \times tr_{\$} + n_{sh} \times sh_{\$}] \tag{51}$$

Where: $f_{\$}$ fuel price, (\$/kg); SC_f fuel consumption specific cost, (\$/t); $\$I_{l\&h}$ loading & hauling capital investments, (M\$); $tr_{\$}$ truck price, (M\$); $sh_{\$}$ shovel price, (M\$); $C_{op.l\&h}$ loading & hauling operating cost, (M\$); n_I number of investment times; and M_{res} ore tonnage reserve, (t).

The total mining specific costs SC_M, (\$/t) is then obtained by:

$$SC_M = SC_{l\&h} + SC_{d\&b} \tag{52}$$

A screenshot for the *Loading and Hauling* sub-model is illustrated in Figure 34.

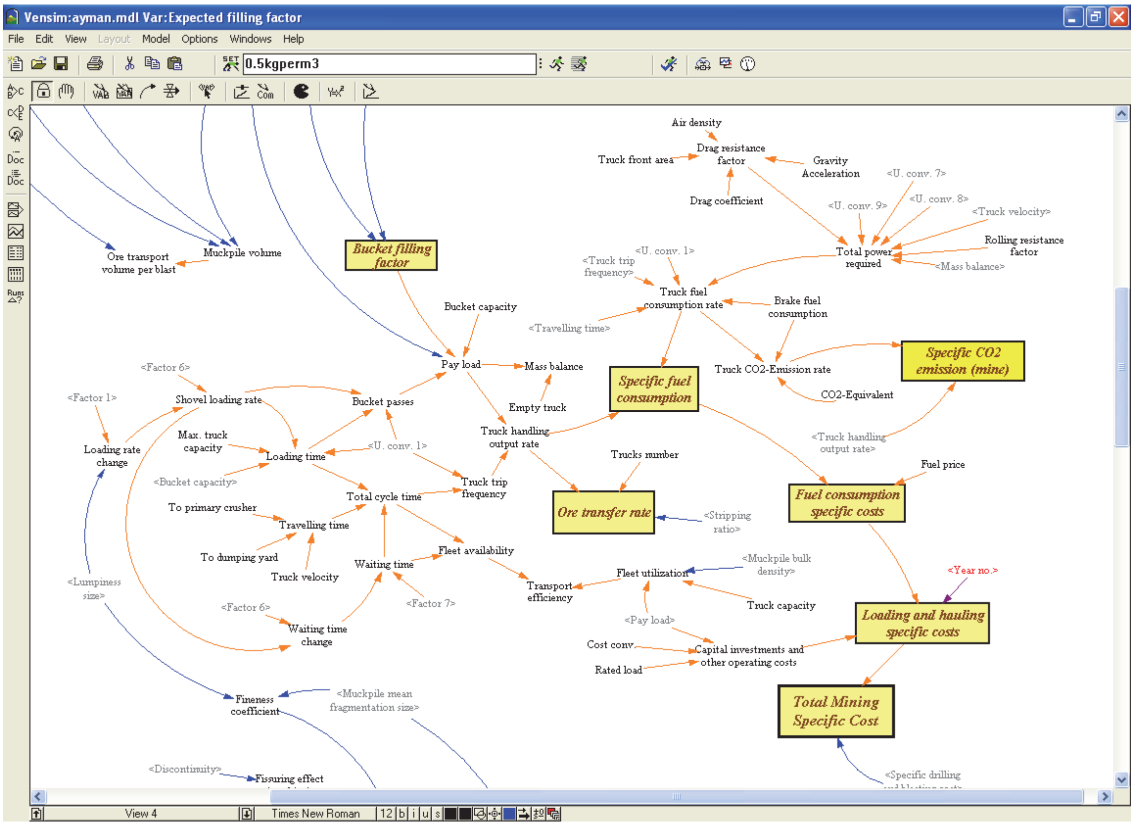


Fig. 34: Screenshot for the *Loading and Hauling* sub-model.

The arrows, which are connecting between the parameters, indicate that there are equations relating between them. The main outputs are included in the yellow rectangles. Also, the arrows, which located in the upper and lower edges of the screenshot, connect certain intermediate parameters to the other sub-models.

The specific CO_2 emission (mine) $Msp._{CO2,M}$, (kg/t) is obtained by:

$$M_{sp.CO2,M} = \frac{\dot{M}_{CO2,tr}}{\dot{M}_{bulk,tr}} \tag{53}$$

$$\dot{M}_{CO2,tr} = \left(\frac{M_{f,trip} \times n_{trip,tr}}{BCF}\right) \times eq_{CO2} \tag{54}$$

Where: $M_{CO2,tr}$ truck CO_2-Emission rate, (kg/h); and eq_{CO2} CO_2-Equivalent, (kg/kWh) [48].

The fleet utilization U_{fleet}, (%); availability γ_{fleet}, (%); and efficiency η_{fleet}, (%) can be estimated as:

$$U_{fleet} = \frac{M_{bulk,tr} \times 100}{n_{trip,tr} \times V_{tr} \times \rho_b} \quad (55)$$

$$\gamma_{fleet} = \frac{(t_{cyc} - t_w) \times 100}{t_{cyc}} \quad (56)$$

$$\eta_{fleet} = \frac{U_{fleet} \times \gamma_{fleet}}{100} \quad (57)$$

4.2.3 Dynamic modeling and simulation for the crushing and grinding operations

Block diagram and data preparation for the inputs and outputs parameters

The block diagram for the sub-model (*Crushing and Grinding*) is shown in Figure 35. The resulted data could be considered as the final output concerning the whole system.

The block diagram gives good information of the final structure of the model and provides a view of how the sub-systems are connected. As it is discussed before, the intermediate and inter-related data are much more than the final outputs of this sub-model.

The uppermost modules indicate the sources of the input parameters which are generated from the previous sub-models, in addition to the available operational, technical, and economical data, while the lowermost modules represent the final output (results).

The intermediate large block shows how and to where the data are directed, in order to be processed within the different model sections. It also includes the much more other intermediate and calculated items.

Processing of the data will be detailed in the next section, which includes the mathematical formulas, lookups and algebraic equations needed for the model construction.

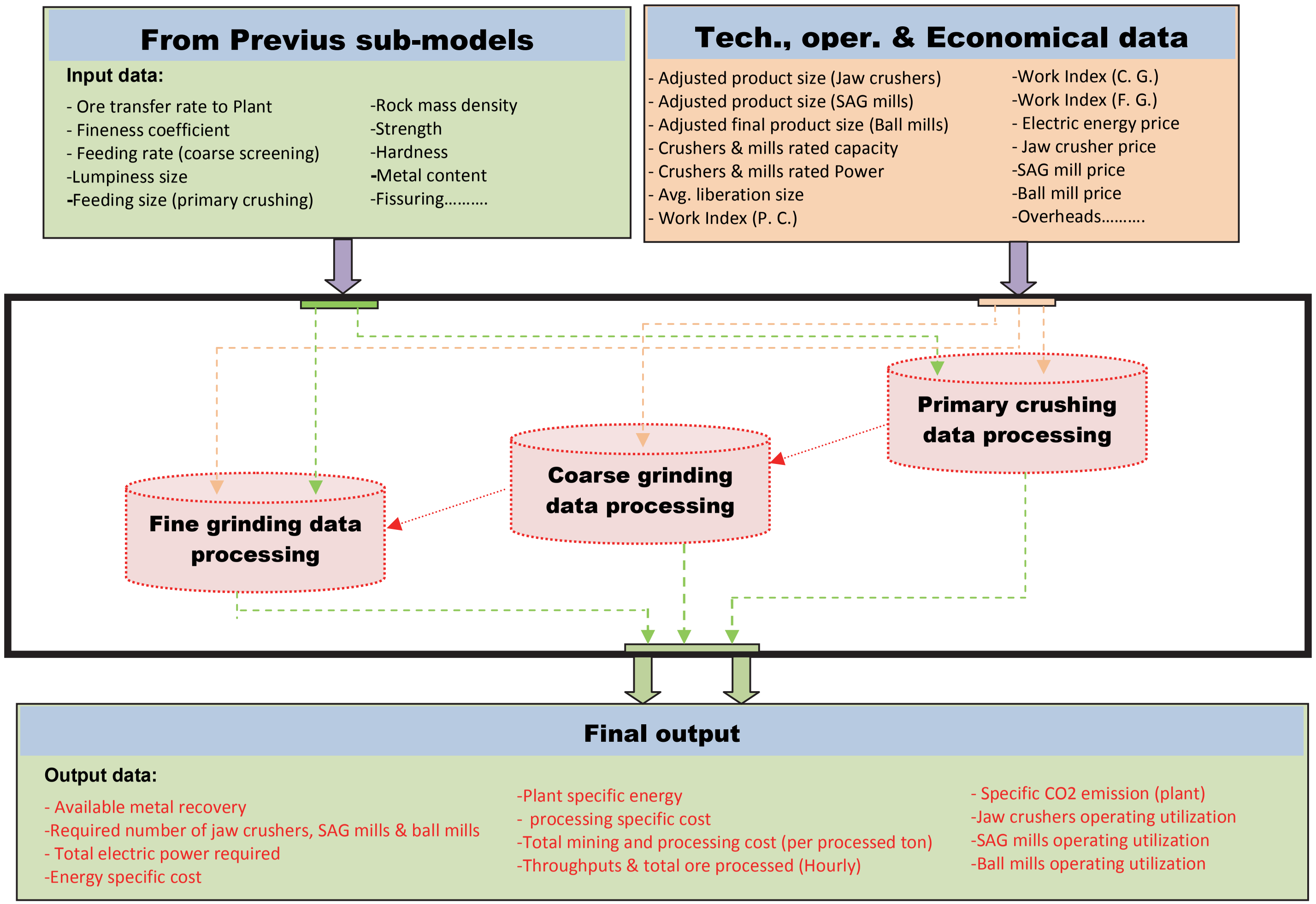


Fig. 35: Block diagram for the data processing within *Crushing and Grinding* sub-model.

The *Crushing and Grinding* sub-model construction

The main goal from this sub-model is to produce a link between the fragmentation size of the blasted rocks, the rock properties and natural parameters, and the delivery rate of ROM to the plant from one side and the final decisions relating to the mine life, the plant facilities, the energy costs, and the sustainability requirements from the other side.

The real image will be reflected by the final total and specific economic and environmental results, which will judge the whole project, such as the total energy required for crushing and grinding, the available metal recovery for the final ground products, the specific cost for the processing operations and the specific greenhouse emission due to the plant machinery and energy consumption. This is in addition to other intermediate results, such as the different stages feeding rates; bypasses; and throughputs, required number of crushers and mills, the plant facilities utilization, the total ore processing rates,….etc.

The following mathematical link all the inputs and the generated intermediate parameters into one net of information, in order to generate finally the output results.

The *Crushing and Grinding* sub-model is divided into three linked sections as follows:

- The primary crushing operation section,
- The coarse grinding operation section, and
- The fine grinding operation section.

a) The primary crushing operation:

The main parameter, which links the current sub-model with the previous one, is the ore transfer rate or the delivery rate of the ROM for the transportation fleet to the plant. The ore transfer rate (to the plant) M_{trans}, (t/h), is calculate as:

$$\dot{M}_{trans} = \frac{\dot{M}_{bulk,tr} \times n_{tr}}{(S_r+1)} \qquad (58)$$

A continuous mining and processing is planned in this model, therefore:

$$\dot{M}_{c.screen} = \dot{M}_{trans} \qquad (59)$$

Where: $M_{c.screen}$ coarse screen feeding rate, (t/h).

The primary crushers feeding rate $M_{p.c}$, (t/h), will be then:

$$\dot{M}_{p.c} = \dot{M}_{c.screen} - \dot{M}_{b.pass,p.c} \quad (60)$$

$$\dot{M}_{b.pass,p.c} = \dot{M}_{c.screen} \times K_{b.pass,p.c} \quad (61)$$

Where: $M_{b.pass,p.c}$ primary crushers bypass, (t/h); and $K_{b.pass,p.c}$ primary crushers bypass factor.

The primary crushers bypass factor is the summation of two designed lookups installed in the model: one represents the fissuring effects (discontinuity and macro fissures from the geological studies of the ore rocks) and the other represents the finesse (the lumpiness size of the blasted muck-pile). Bypassing of the primary crushers to the next stage will be done across a coarse screen.

Natural macro-fissures of the ore-deposit rock mass and the micro-fissures, which are introduced to the ROM fragmentation by means of blasting, are explained before in detail, (Appendix 1). These two important physical characteristics, Figure 36, have a considerable effect on the degree and efficiency of the primary crushing.

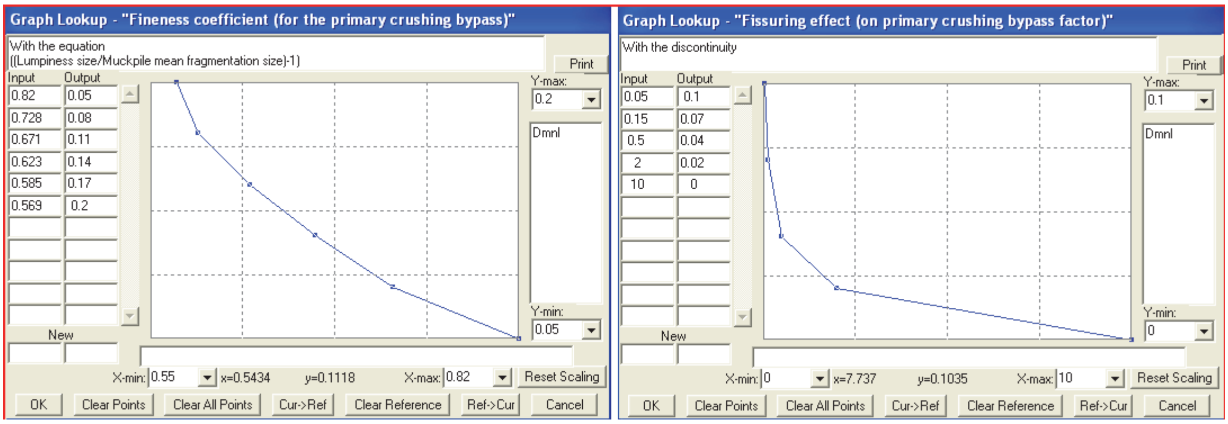
This is also reacted with the effect of the mechanical operations of loading and hauling on the fineness and the final fragmentation feeding size to the plant.

By applying *Bond's Equation* for the electric power requirements [14], in due to rock fragmentation and size reduction, we can obtain the following:

$$P_{p.c} = \frac{1}{10} \times WI_{p.c} \times \left[pX_{p.c}{}^{-0.5} - fX_{p.c}{}^{-0.5} \right] \times \dot{M}_{p.c} \quad (62)$$

Where: $P_{p.c}$ required power for primary crushing; $WI_{p.c}$ primary crushing work index, (kWh/t);

$pX_{p.c}$ 80% passing size of the primary crushing product, (µm); and $fX_{p.c}$ 80% passing size of the primary crushing feed, (µm).



(a) (b)

Fig. 36: A screenshot for an example of two lookups within the 3rd sub-model:
(a) The fineness coefficient (crushing); (b) The fissuring effect (crushing).

The primary crushers feeding size, is considered, here in the model, as 80% of the lumpiness size, which was calculated from the previous mentioned Rosin-Rammler equation, as a result of further fragmentation during the loading and hauling operation.

Afterwards, for a given suitable and specific Jaw Crusher type, the real number, which should be in duty, can be obtained from:

$$n_{J.c,real} = \left[n_{J.c,theo} \right] \tag{63}$$

$$n_{J.c,theo} = MAX\left\{ \frac{P_{p.c}}{RP_{J.c}} : \frac{\dot{M}_{p.c}}{R\dot{M}_{J.c}} \right\} \tag{64}$$

Where: $n_{J.c,real}$ number of *Jaw Crushers* in duty; $n_{J.c,theo}$ theoretical required number of *Jaw Crushers*; $RP_{J.c}$ *Jaw Crusher* rated power, (kW); and $RM_{J.c}$ *Jaw Crusher* rated capacity, (t/h).

Then, the primary crushing facility operating utilization $U_{p.c}$, (%), can be obtained by:

$$U_{p.c} = MAX\left\{ \frac{\dot{M}_{p.c}}{R\dot{M}_{J.c} \times n_{J.c,real}} : \frac{P_{p.c}}{P_{avail,p.c}} \right\} \tag{65}$$

$$P_{avail,p.c} = RP_{J.c} \times n_{J.c,real} \tag{66}$$

Where: $P_{avail,p.c}$ primary crushing available power, (kW).

As a constraint within primary crushing, the feeding size $fX_{p.c}$ should be ≤ 75% of the crusher mouth diagonal [67], and the primary crushers' throughput is equal to their feeding rate.

b) The coarse grinding operation:

The subsequent coarse grinding operation is considered as a natural extension for the previous one that:

$$\dot{M}_{s.sceen} = \dot{M}_{thr,p.c} \tag{67}$$

Where: $M_{thr,p.c}$ primary crushers' throughput, (t/h); and $M_{s.screen}$ secondary screen feeding rate, (t/h).

The coarse grinding feeding rate $M_{c.g}$, (t/h), will be then:

$$\dot{M}_{c.g} = \dot{M}_{s.sceen} + \dot{M}_{b.pass,p.c} - \dot{M}_{b.pass,c.g} \tag{68}$$

$$\dot{M}_{b.pass,c.g} = \dot{M}_{s.sceen} \times K_{b.pass,c.g} \tag{69}$$

Where: $M_{b.pass,c.g}$ coarse grinding bypass, (t/h); and $K_{b.pass,c.g}$ coarse grinding bypass factor.

The coarse grinding bypass factor is represented within the model by a lookup, which relates it with the previous primary crushing bypass.

By further application to *Bond's Equation* for the electric power requirements in due to rock fragmentation and size reduction, we can obtain the following:

$$P_{c.g} = \frac{1}{10} \times WI_{c.g} \times \left[pX_{c.g}{}^{-0.5} - fX_{c.g}{}^{-0.5}\right] \times \dot{M}_{c.g} \tag{70}$$

$$fX_{c.g} = pX_{p.c} \tag{71}$$

Where: $P_{c.g}$ required power for coarse grinding, (kW); $WI_{c.g}$ coarse grinding work index, (kWh/t); $pX_{c.g}$ 80% passing size of the coarse grinding product, (µm); and $fX_{c.g}$ 80% passing size of the coarse grinding feed, (µm).

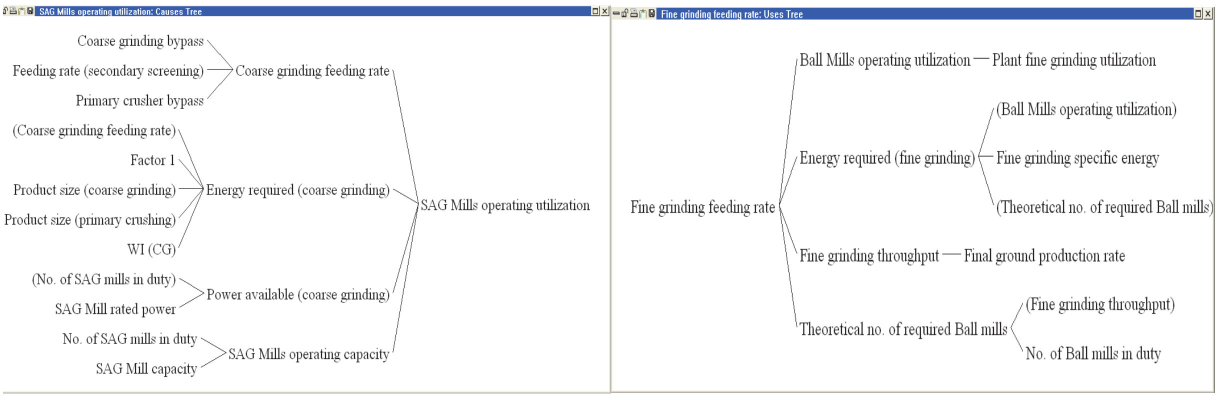
Afterwards, for a given suitable and specific SAG Mill type, the real number, which should be in duty, can be obtained from:

$$n_{S.m,real} = \lceil n_{S.m,theo} \rceil \quad (72)$$

$$n_{S.m,theo} = MAX\left\{\frac{P_{c.g}}{RP_{S.m}} : \frac{\dot{M}_{c.g}}{R\dot{M}_{S.m}}\right\} \quad (73)$$

Where: $n_{S.m,real}$ number of *SAG Mills* in duty; $n_{S.m,theo}$ theoretical required number of *SAG Mills*; $RP_{S.m}$ *SAG Mill* rated power, (kW); and $RM_{S.m}$ *SAG Mill* capacity, (t/h).

The tools of (uses) and the (causes) tree make a good tracing for the different inter-related parameters, in order to trace weakness, errors, units unbalance or any other modeling warnings. An example for a uses tree and a causes tree for two different parameters is shown in Figure 37.



(a) (b)

Fig. 37: A screenshot illustrates a (uses) and a (causes) tree for two related parameters.
(a) The causes tree for the SAG mills utilization; (b) The uses tree for the fine grinding feeding rate.

Then, the coarse grinding facility operating utilization $U_{c.g}$, (%), can be obtained by:

$$U_{c.g} = MAX\left\{\frac{\dot{M}_{c.g}}{R\dot{M}_{S.m} \times n_{S.m,real}} : \frac{P_{c.g}}{P_{avail,c.g}}\right\} \tag{74}$$

$$P_{avail,c.g} = RP_{S.m} \times n_{S.m,real} \tag{75}$$

Where: $P_{avail,c.g}$ coarse grinding available power, (kW).

As a constraint within coarse grinding, the coarse grinding throughput for the SAG mills is equal to their feeding rate.

c) The fine grinding operation:

The subsequent fine grinding operation is considered also as a natural extension for the previous coarse grinding one that:

$$\dot{M}_{c.c} = \dot{M}_{thr,c.g} \tag{76}$$

Where: $\dot{M}_{c.c}$ cyclone cluster feeding rate, (t/h); and $\dot{M}_{thr,c.g}$ coarse grinding throughput, (t/h).

For the purpose of not repetition, all the equations from (88) to (95) are the same by using the corresponding parameters belonging to the fine grinding process.

It should be noticed that the fine grinding bypass factor is designed in the model by installation of two lookups: one represents the hardness effects (as a function to the rock strength) and the other is to relate it with the previous coarse grinding bypass.

As a constraint within fine grinding, the fine grinding throughput for the Ball mills is equal to their feeding rate.

As a second important constraint, belonging to the processed ore mass transfer for grantee of the continuous mining and processing, the final milled ore tonnage rate should be approximately equal to the ore transfer rate to the plant, i.e.:

$$\dot{M}_{mill} \cong \dot{M}_{trans} \quad (77)$$

$$\dot{M}_{mill} = \dot{M}_{thr,f.g} + \dot{M}_{b.pass,f.g} \quad (78)$$

Where: $M_{b.pass,f.g}$ fine grinding bypass, (t/h); $M_{thr,f.g}$ fine grinding throughput, (t/h); and M_{mill} total milled ore tonnage rate, (t/h).

Thereafter, the total plant specific energy requirements SE_{plant},(kWh/t), can be then calculated from:

$$SE_{Plant} = SE_{p.c} + SE_{c.g} + SE_{f.g} \quad (79)$$

$$SE_{p.c} = \frac{P_{p.c}}{\dot{M}_{mill}} \quad (80)$$

$$SE_{c.g} = \frac{P_{c.g}}{\dot{M}_{mill}} \quad (81)$$

$$SE_{f.g} = \frac{P_{f.g}}{\dot{M}_{mill}} \quad (82)$$

Where: SE_{Plant} plant specific energy, (kWh/t); $SE_{p.c}$ primary crushing specific energy, (kWh/t); $SE_{c.g}$ coarse grinding specific energy, (kWh/t); and $SE_{f.g}$ fine grinding specific energy, (kWh/t).

Then, the processing specific costs SC_P, (\$/t), can be obtained as:

$$SC_P = \left(\frac{\$I_{c\&g} + C_{op.c\&g}}{M_{res}}\right) + SC_{energy} \quad (83)$$

$$SC_{energy} = Elec_{\$} \times SE_{Plant} \quad (84)$$

$$\$I_{c\&g} = n_I \times \left[n_{J.c,real} \times JC_{\$} + n_{S.m,real} \times SM_{\$} + n_{B.m,real} \times BM_{\$}\right] \quad (85)$$

Where: $Elec_{\$}$ electric energy price, (\$/kWh); SC_{energy} electric energy consumption specific cost, (\$/t); $\$I_{c\&g}$ crushing & grinding capital investments, (M\$); $JC_{\$}$ jaw crusher price, (M\$); $SM_{\$}$ SAG mill price, (M\$); $BM_{\$}$ ball mill price, (M\$); and $C_{op.c\&g}$ crushing & grinding operating cost, (M\$).

The total mining and processing costs per processed tonnage $SC_{m\&p}$, (\$/t), is then obtained by:

$$SC_{M\&P} = SC_P + SC_M \times (S_r + 1) + SC_{f\&c} \tag{86}$$

Where: $SC_{f\&c}$ floatation and other concentration specific costs, (\$/t).

The specific CO_2 emission (processing) $Msp._{CO2,P}$, (kg/t), is obtained as:

$$M_{sp.CO2,P} = SE_{Plant} \times eq_{CO2} \tag{87}$$

As it is planed to be a continuous mining and processing, the mine life n_y, (Years), can be estimated as:

$$n_y = \frac{M_{res}}{\dot{M}_{mill} \times n_d \times n_h} \tag{88}$$

A screenshot for the Crushing and Grinding sub-model is illustrated in Figure 38. The arrows, which are connecting between the parameters, indicate that there are equations relating between them. Some of the main outputs are included in the yellow rectangles, while many others, which are intermediate and also independent final results, are located out of the figure boundaries.

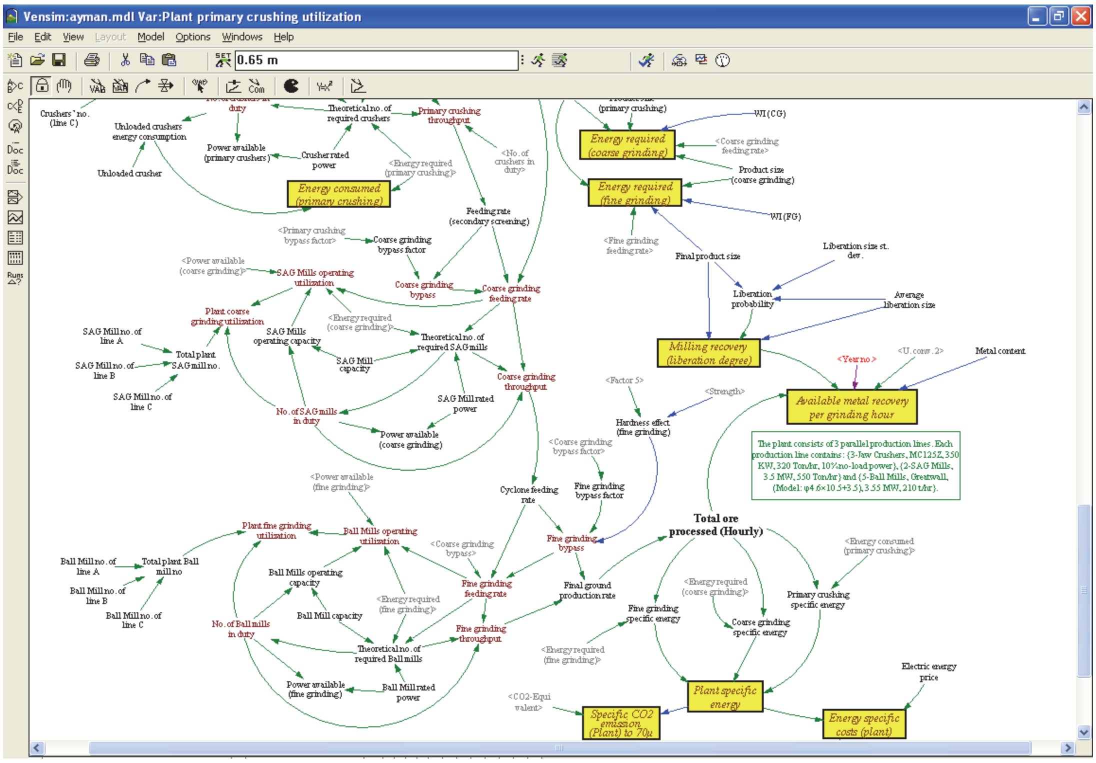


Fig. 38: Screenshot for a part of the *Crushing and Grinding* sub-model.

5. Case Study Application and the Model Output and Assessment

5.1 Main physical properties of the ore deposit under study

A case study data, which are generated from a real copper open pit mine with some other more assumptions for the missed information, are considered here in this thesis, as mentioned before, (see chapter 3). Table 7 shows the general physical characteristics for the under investigation ore deposit. This porphyry copper deposit is consisting of three principal ore types, which are localized in three adjacent locations within the mineralization area.

Table 7: The general physical characteristics for the under investigation ore deposit.

Item	Unit	Ore Type (A)	Ore Type (B)	Ore Type (C)	Wt. Avg.
Existence contribution	%	43	27	30	-
Rock mass strength	MPa	95	130	148	120
Discontinuity	m	0.7	1.2	2	1.23
Drilling penetration rate	m/h	40	30	17	26.8
Primary crushing (WI)	kWh/t	13	14	18	14.8
Coarse grinding (WI)	kWh/t	14	15	19	15.8
Fine grinding (WI)	kWh/t	15	16	20	16.8
Metal content	%	0.9	1.0	0.8	0.89
Avg. liberation size	µm	250	200	165	211
Final ground size (for 95% rec.)	µm	185	100	70	127

They mainly differ in their included copper mineralization resource and metal content, their rock mass strength, their mineral liberation grain size and their grain size distribution configuration.

They are also different in their rock types work indices (WI). Moreover, they have different existence distributions and magnitudes (contributions) within the mineralization area.

It is assumed that the specific weight for the ore rocks and the overburden rocks are the same and equals to 2.5 t/m^3, while within the model, it will be a range for changing all the inputs concerning the ore physical properties or the other technological and operational factors.

5.2 Principal technological and operational parameters within the case study

Project main requirements, criteria and availabilities

The total ore deposit reserves are 200 Mt. Hence, they are distributed, according to the contribution ratios in the previous table, to 86 Mt of the ore type (A), 54 Mt of the ore type (B) and 60 Mt of the ore type (C).

The minimum required annual productivity is 10 Mt (Ore), under a stripping ratio of 2:1. Therefore, the total displaced ore and overburden of the whole deposit, which is used in the primarily choice of the loading and transporting facilities, will be equal to 600 Mt.

Drilling and blasting technology is used for the rock mass loosening and displacement, while there is no possibility for post or secondary blasting for the boulders, if encountered.

According to the planed annual productivity, the maximum plant capacity and other environmental and organizing constraints, it is assumed to achieve blasting, one time every three days, while the drilling operations are continued for the full working hours time.

The mining activities (loading and hauling) are performed through the total available working hours of 5,400 h/y, which are calculated as the net working time due to weathering, emergency, maintenance, and other operational considerations.

The primarily assumed mine life, according to the minimum required annual productivity and the available reserves is about twenty years, which will be used to build the reference mode of the model and will be adapted according to the first modeling results and outputs. This adaptation will have certain constraints, which will be assigned in the next coming sections, in order to maintain the possibility for online mining and processing. This will have special benefits in the case of the model further optimization for the different production-lines configurations and the online ore blending strategies.

Technological and operational parameters within drilling and blasting stage

Drilling and blasting data are planed, according to a blasting strategy, to suit the required annual productivity with the criteria of the available loading and transporting fleet and the plant maximum capacity to achieve, primary, the online continuity in mining and processing in the first reference mode, before further optimizations, for the global optimization with economical, environmental, marketing, and sustainable considerations.

From the main assumed data for the drilling and blasting estimations is the borehole diameter, which is chosen according to the used explosive type, the rock characteristics, the available drilling rigs and the other blasting pattern parameters. Three PV-351 drilling-rigs from Atlas Copco Blasthole Drills [8] are assumed to be operated 22 hours a day within the mining area. The borehole diameter is 0.27 m (10.5"), while the bench height is 15 m. The calculated blasted tonnage is 300,000 t/blast by using emulsion explosive of 80 % ANFO and 20 % high explosive (Emulsion).

Technological and operational parameters within loading and hauling stage

According to the required annual productivity, the maximum plant capacity, the hauling distance, the average trucks velocity ….etc, two suitable extraction and transportation strategies are investigated, regarding to the Loading and Hauling sub-model.

The 1st strategy (Fleet A) investigates the utility of 3-hydraulic shovels 6040 FS [19], 22 m^3, and 12 dumping trucks 789C [21], 177 t, 105-120 m^3 (heaped capacity). While the 2nd strategy (Fleet B) investigates the utility of 2-hydraulic shovels 6060 FS [20], 34 m^3, and 8 dumping trucks 793F [22], 227 t, 160-190 m^3 (heaped capacity). These fleets are products from Caterpillar Inc. and concluded in Table 8.

Capital investments and other operating costs for the loading and hauling sub-model, according to the fleet strategy, contain ownership, taxes, insurance ... and other operating costs.

Table 8: The main characteristics for the two loading and hauling strategies.

Loading and hauling strategy/Item	Shovels		Trucks	
	No.	Capacity	No.	Capacity
The 1st strategy (Fleet A)	3	6040 FS, 22 m^3	12	789 C, 177 t, 105-120 m^3
The 2nd strategy (Fleet B)	2	6060 FS, 34 m^3	8	793 F, 227 t, 160-190 m^3

The calculated operating costs are changed after a certain duty limit according to the transferred pay load magnitude. The transportation system have a maximum operating time (life time) of 60,000 hours, and hence, for the 20 years project, two investment times are planed for the model.

The fragmented muck-pile is loaded to the trucks by means of the hydraulic shovels and hauled to the dumping yard or to the plant, according to the payload type. The dump distance is assumed to be equal to the plant distance which is 1500 m from the extraction area.

Technological and operational aspects within crushing and grinding stage

The plant consists of 3-parallel production lines, each of which can process ROM rate up to 1000 t/h, to achieve a total plant maximum capacity of 3,000 t/h.

Each production line contains the followings: 3-Jaw crushers (Primary Crushing), 350 kW, 320 t/h [69]; 2-SAG mills (Coarse Grinding), 3.5 MW, 550 t/h [73]; and 5-Ball mills (Fine Grinding), 3.55 MW, 210 t/h [140], which are concluded, with their technical grain size limits, in Figure 39.

The primary crushing product size is adjusted to 10 cm, while for the coarse grinding it is adjusted to 1 mm, and for the fine grinding (final product size) is adjusted, as a first concept, to 70 µm, which is the minimum liberation grain size all over the ore deposit, as detailed before in Table 7.

Capital investments and other operating costs for the crushing and grinding sub-model contain ownership, taxes, constructions… and other operating costs. The calculated operating costs are 10 % of the investment capital cost for the three production lines system of a total capacity of 3,000 t/h. The maximum operating time (life time) for the plant facilities is 10 years [140], and hence, for the 20 years project, two investment times are planed for the model.

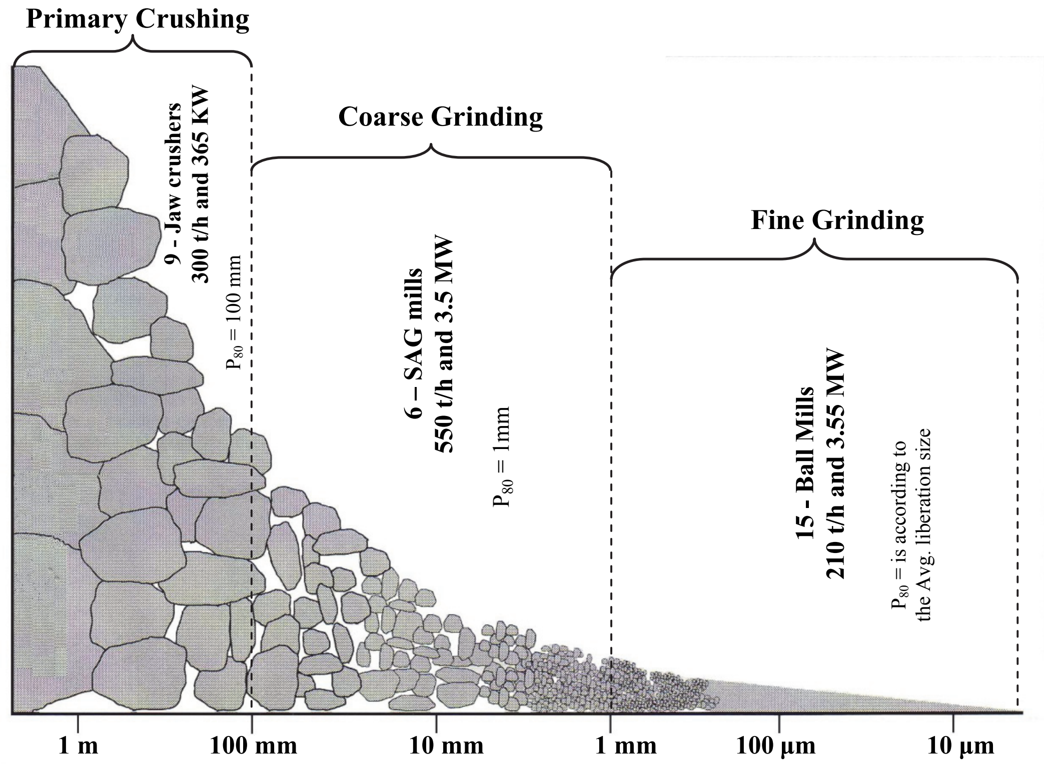


Fig. 39: The available plant main facilities.

5.3 Processing of the data from the case study

The available data of the case study is tabled, prepared and arranged. This includes data, which belong to the different facilities specifications, such as technical specifications of the drilling rigs, hydraulic shovels, dumping trucks, primary crushers, SAG mills, Ball mills, etc, which are quoted from their technical catalogs, manuals or websites.

The other missed information is detailed from other referenced mining projects, which alike the investigated copper-porphyry case-study. This includes different operational parameters and average prices such as the drilling fixed costs [7, 8], explosive price [47, 93, 116], fuel price [49, 51, 138, 144], electric energy price [138], metal price [99, 107, 108, 121, 141], CO_2-Equivalent [48], brake fuel consumption (BFC) [21, 42, 136, 137], Crushers and mills prices [69, 73, 105], and renting price for the other components, which are not involved within the capital assets [7]. Table 9 shows the most important assumed and quoted data for the case-study.

Table 9: Important referenced assumed and quoted data for the case-study.

Item	Unit	Value	Item	Unit	Value
Fuel (Diesel) price	$/L	1.25	**Crusher price**	M$	0.4
Electric price	$/kWh	0.17	**SAG mill price**	M$	3
Metal-Cu price	$/t ($/lb)	5500 (2.5)	**Ball mill price**	M$	1
ANFO explosive price	$/kg	0.4	**Brake fuel consumption**	Kg/kWh	0.28
Emulsion explosive price	$/kg	1.2	**Discount (Interest) rate**	%	15

5.4 [*Reference-Mode*] model results and assessment

The main natural parameters, which belong to the ore deposit, are fed as inputs to the model. In addition to the main inputs regarding the operational and technical parameters, which belonging to the drilling, blasting, loading, hauling, crushing, and coarse and fine grinding, other individual model runs are designed, in order to examine the different outputs possibilities.

5.4.1 Preliminary main results of the mining activities sub-models

Starting with a current state (State A) adjusted to a mean blast fragmentation size (X_{50}) of 40 cm, a number of different fragmentation size (X_{50})s between 65 cm and 15 cm are investigated dynamically through different model runs. This will help in choosing the fragmentation size range and, hence, the corresponding powder factors range, at which their effects on the different subsequent operations costs and efficiencies would be further investigated.

The chosen range of the specific explosive energies is (powder factors) investigated for the two loading and hauling fleet configurations, according to the previously stated considerations (section 5.2), such as the plant capacity, and also the mining specific cost.

By other technical, economical, environmental, and sustainability considerations and constraints, the life of the project will be chosen, in addition to the optimal fragmentation size and the best fleet configuration, in order to be used in the suggested further model optimization in the next chapter.

Specific explosive energy range choice according to the mining activities constraints

The most important criteria here in choosing the range of fragmentation size for the further investigation will be as follows:

- Bucket filling factor should not be less than 55 %, in order to suit the muck-pile extraction by the different available surface mining technologies, as below this value is suiting to other technologies such as marble and granite quarries, but not the blasted ore deposit fragmentation.
- Trucks maximum allowable pay load should not be more than 110 % of its rated pay load.
- The ore delivery rate to the plant should not exceed the plant capacity (3,000 t/h).
- According to the primary crusher mouth and that secondary blasting is not allowable, lumpiness should not exceed 90 cm [67].
- The range of the least total mining specific cost is considered also an important criterion for the range choice.

Table 10 shows the results for the model runs due to the experimented different blast fragmentation sizes. In the table, each column is considered an individual separated simulation run. It should be mentioned that, these are not all the outputs generated from the runs, but just those, which are suitable here to illustrate the upper stated criteria for the choice of the new narrow and suitable practicable range of values for the further investigations.

The orange colored cells in the table indicate the limits for the allowable values for the previously mentioned constraints, while the red dashed rectangle represents the boundaries for the choice. The two values with the red font are corresponding to the range of the powder factor, which will be further expanded for the deeply next investigations for the global mining and processing operations, after choosing of the suitable transportation fleet.

Table 10: Fragmentation size model runs, results and criteria for choice of the suitable practicable range.

Item/Run	Unit	1	2	3	4	5	6	7	8	9	10	11
Mean fragment size	m	**0.15**	**0.20**	**0.25**	**0.30**	**0.35**	**0.40**	**0.45**	**0.50**	**0.55**	**0.60**	**0.65**
Spec. explosive energy	kg/m^3	3.23	2.05 1.5	1.44	1.08	0.85	0.69	0.57	0.48 0.5	0.41	0.36	0.32
Spec. drilling & blasting cost	$/t	0.845	0.586	0.452	0.372	0.321	0.286	0.260	0.241	0.226	0.214	0.205
Lumpiness size	m	0.23	0.31	0.40	0.49	0.59	0.70	0.80	0.91	~~1.03~~	~~1.15~~	~~1.28~~
Bucket filling factor	Dmnl	0.92	0.87	0.825	0.76	0.69	0.61	0.60	0.59	0.58	0.56	0.55
Bulk density	t/m^3	2.25	2.14	2.03	1.87	1.69	1.53	1.50	1.47	1.41	1.37	1.34
Ore delivery rate	t/h	~~3655.63~~	~~3281.83~~	2775.15	2350.70	1930.48	1477.84	1406.90	1303.15	1230.91	1165.76	1014.10
Specific fuel consumption	kg/t	0.386	0.402	0.420	0.452	0.496	0.556	0.571	0.580	0.600	0.620	0.638
Fleet availability	%	92	92	92	91	90	90	89	89	89	88	88
Fleet utilization	%	~~106~~	100	94	91	82	73	71	70	69	68	63
Transport efficiency	%	98	92	86	83	74	66	63	62	61	60	55
Loading and hauling spec. cost	$/t	1.222	1.182	1.134	1.120	1.171	1.240	1.257	1.268	1.290	1.313	1.334
Total mining specific cost	$/t	2.067	1.768	1.586	1.492	1.492	1.525	1.517	1.508	1.516	1.527	1.539

It should be mentioned that the run no. 6 is the current run or, in other words, the assumed initial state (State A) of the suggested fragmentation size (0.4 m).The range for the accepted powder factor (specific explosive energy) is found to be between 0.5 and 1.5 kg/m^3.

Mining activities economical and environmental main results

Eleven runs are made within the chosen range of the specific explosive energy, which is between 0.5 and 1.5 kg/m^3 with a step of 0.1 kg/m^3. The collected total results are presented in Table 11.

Mining activity results (drilling-to-hauling), which are presented in this table are due to the applying of the fleet strategy (Fleet A), while the corresponding data output for (Fleet B) are presented in the appendices, Table Ap2-1.

The blasting main economic and operational results are presented in Figure 40. As illustrated in the figure, the muck pile mean fragmentation sizes corresponding to the under investigation specific explosive energy span are drawn with the respective bucket filling factors and also the resultant specific drilling and blasting cost.

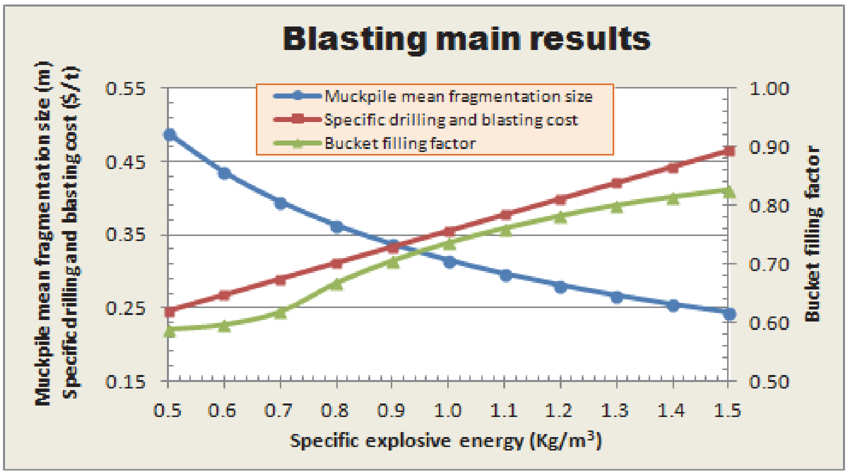


Fig. 40: Blasting main economic and operational results.

It is noticed that, by changing the powder factor from 0.5 to 1.5 kg/m^3, the bucket filling factor state is modified from 59 % to 84 %. This is happened due to the modification of the mean blast fragmentation size from 49 cm to 23 cm.

Table 11: Mining activity results (drilling-to-hauling) for the specific explosive energy model runs.

Item/Run	**Unit**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Spec. explosive energy	kg/m^3	**0.50**	**0.60**	**0.70**	**0.80**	**0.90**	**1.00**	**1.10**	**1.20**	**1.30**	**1.40**	**1.50**
Mean fragment size	m	0.49	0.44	0.39	0.36	0.34	0.31	0.30	0.28	0.27	0.25	0.24
Spec. drilling & blasting cost	$/t	0.245	0.267	0.289	0.311	0.333	0.355	0.377	0.399	0.421	0.443	0.465
Bucket filling factor	Dmnl	0.59	0.60	0.62	0.67	0.70	0.74	0.76	0.78	0.80	0.81	0.83
Pay load	t	114.78	118.19	126.20	145.03	161.37	175.78	188.64	200.23	210.76	217.33	223.00
Ore delivery rate	t/h	1311.82	1418.29	1514.42	1740.33	2038.33	2220.33	2382.77	2529.18	2662.21	2745.28	2816.87
Total cycle time	Min	21	20	20	20	19	19	19	19	19	19	19
Truck fuel consumption rate	kg/h	189.64	201.66	207.62	221.62	246.07	257.35	267.42	276.49	284.74	289.88	294.32
Specific fuel consumption	kg/t	0.578	0.569	0.548	0.509	0.483	0.464	0.449	0.437	0.428	0.422	0.418
Fuel consumption specific costs	$/t	0.665	0.654	0.631	0.586	0.555	0.533	0.516	0.503	0.492	0.486	0.481
Truck CO_2-emission rate	kg/h	71.79	76.34	78.60	83.90	93.16	97.43	101.24	104.67	107.79	109.74	111.42
Specific CO_2 Emission (mine)	kg/t	0.219	0.215	0.208	0.193	0.183	0.176	0.170	0.166	0.162	0.160	0.158
Fleet availability	%	89	90	90	91	91	91	91	91	92	92	92
Fleet utilization	%	67	68	71	77	81	84	87	90	92	93	95
Transport efficiency	%	60	61	64	70	74	76	79	82	85	86	87
Loading and hauling spec. cost	$/t	1.265	1.254	1.231	1.186	1.155	1.133	1.116	1.103	1.116	1.129	1.141
Total mining specific cost	$/t	1.510	1.521	1.520	1.497	1.488	1.488	1.493	1.502	1.537	1.572	1.606
Project life	Year	28.2	26.1	24.5	21.3	18.2	16.7	15.5	14.6	13.9	13.5	13.1

By reducing the fragments size, the loading machine efficiency will be better and the capability of good filling and compacting the material will be also modified as well as the changing in the loading time. In the same time, the further fragmentation size reduction is accompanied with more explosive quantities in each borehole and, in some cases, more drilling pattern narrowing with more drilling operating costs. This is the cause of increasing the corresponding drilling and blasting cost, as seen in the figure.

The muck-pile mass transfer rates by the dumping trucks, with their average hauled pay loads, are illustrated in Figure 41. The individual truck pay load is increasing by increasing of the loading machine bucket filling factor, which is already modified by increasing the explosive specific energy. The more reducing in the fragmentation size, the less the voids between the loaded rocks will be and, hence, this results in less swelling and better ore compaction within the truck. Accordingly, and as it is shown in the figure, this increasing in the hauled pay load for each dumping truck will result in increasing of the delivery rate of the ROM to the primary crushers.

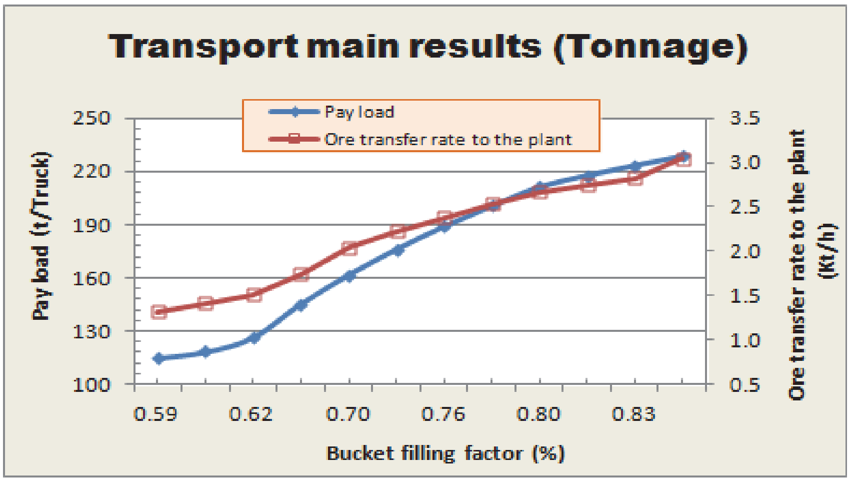


Fig. 41: Mass transfer rate to the plant.

The truck fuel consumption rate and the corresponding fuel consumption specific costs are drawn versus the resultant bucket filling factors and illustrated in Figure 42. The fuel consumption of the individual truck increases by increasing its pay load.

This is due to the need for the more power requirements to offset the resultant more resistance of the total truck mass balance. But it should be mentioned that the overall (the total fleet) fuel consumption specific cost is decreasing, as it is illustrated in the figure. This is due to the higher ROM delivery rate to the plant.

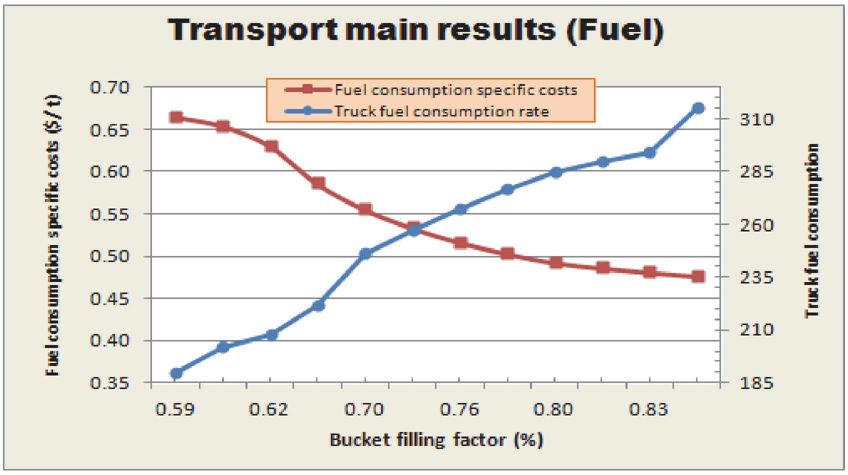


Fig. 42: Fuel consumption rates and costs.

The counteractive environmental resultant aspects, due to loading and hauling of the ore deposit fragmentations, are considered here in the reference mode of the model as merely due to the CO_2 emissions of the diesel powered machinery, such as the dumping trucks.

The individual truck CO_2 emission rate, the total CO_2 tonnage emitted, and the transportation efficiency are drawn versus the bucket filling factors and illustrated in Figure 43.

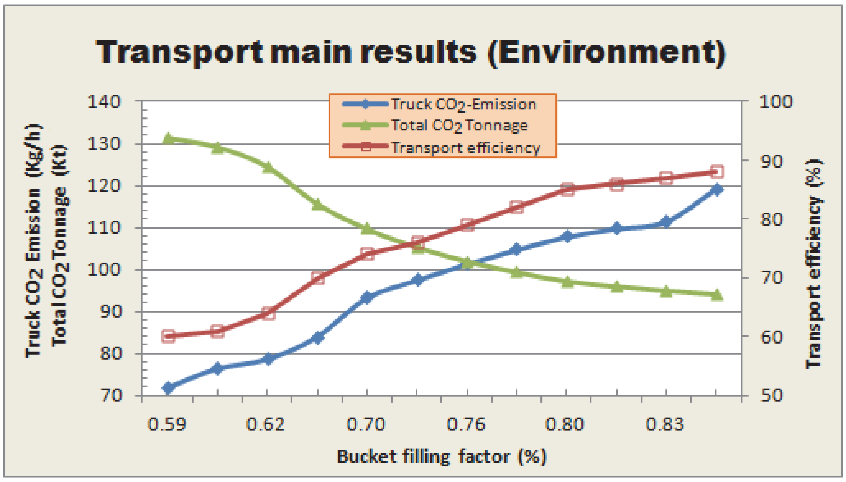


Fig. 43: Total CO_2 emission and the transport efficiency.

As seen from the figure, the CO_2 emission rate of the dumping truck increases by increasing the bucket filling factor and this is due to the more power requirements and the accompanied higher fuel consumption rates. Due to improvement of the loading and waiting times by the better and faster loading of the trucks, due to the better fragmentations, the efficiency of the whole loading and hauling operation is improved from 60 % to 88 %, across the investigated span. It should be mentioned here also that, as shown in the figure, the final total produced tonnage of the emitted CO_2 is decreasing, across the investigated span, due the decreasing in the overall project life, as will be explained in a next paragraph.

The total mining specific cost, which is the summation of the drilling and blasting costs and the loading and hauling costs, is illustrated with its components in Figure 44. It is clearly shown in the figure, how the reduction in the loading and hauling cost, due to the previously mentioned reasons, mitigates the increasing in the drilling and blasting cost, which results from the required more specific explosive energy. This happens until a certain limit, at which the increasing in the drilling and blasting costs and the increasing in the operational and maintenance expenses of the fleet, overcome this action.

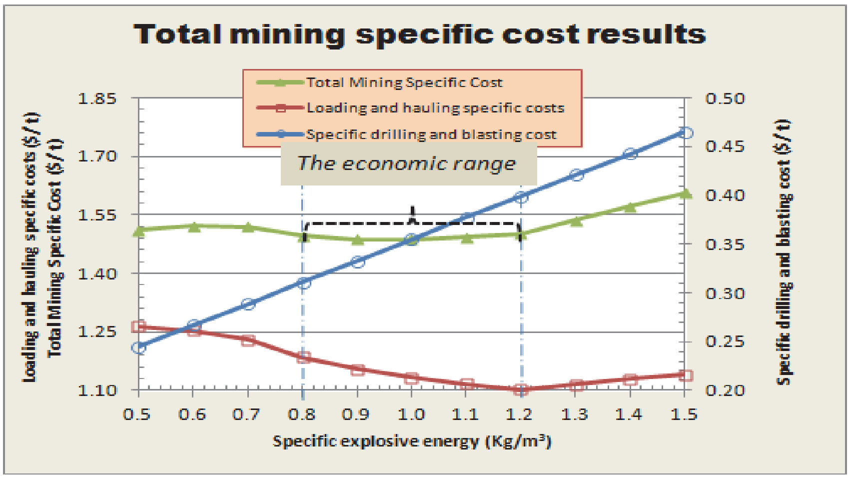


Fig. 44: Total mining costs and the best economic range.

Thus, an area of the best specific explosive energy range, which is between 0.8 and 1.2 kg/m^3 and is corresponding to the least total mining specific costs, can be indicated, as shown in the figure, by the dash-doted blue lines. From this economic range and by referring to Table 11, it can be seen that it is corresponding to a fragmentation size between 28 and 36 cm.

Figure 45 is showing that (Fleet A) has always lower specific costs than (Fleet B), across the span of the fragmentation size under investigation. This is especially within the previously mentioned economic range.

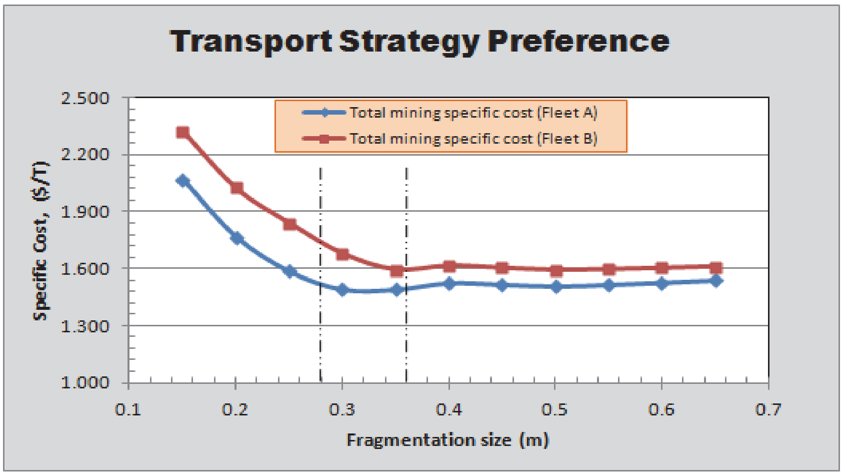


Fig. 45: Loading and hauling strategies comparison and preference.

Mine life range according to the mining and processing activities constraints

In this study, some constraints are assumed, in order to help in ranging the planed possible life time of the project. Some of them are belonging to the copper metal market, such as the general situation of the market demand and the ore reserves availabilities. Other constraints are belonging to environmental and sustainability concepts and the rights of the intra-generational equity, as mentioned before, (Ch.1), in addition to the project technical capacities and constraints.

The assumed constraints for the life time range can be concluded as:

- The total ore reserves, (200 Mt);
- The minimum annual productivity, (10 Mt, Ore);
- The maximum plant capacity, (3,000 t/h);
- Economic and technical planning requirements, (not more than 20 years);
- Sustainability and marketing concepts, (not less than 12 years); and
- The previously determined economic range, (0.8 to 1.2 kg/m^3).

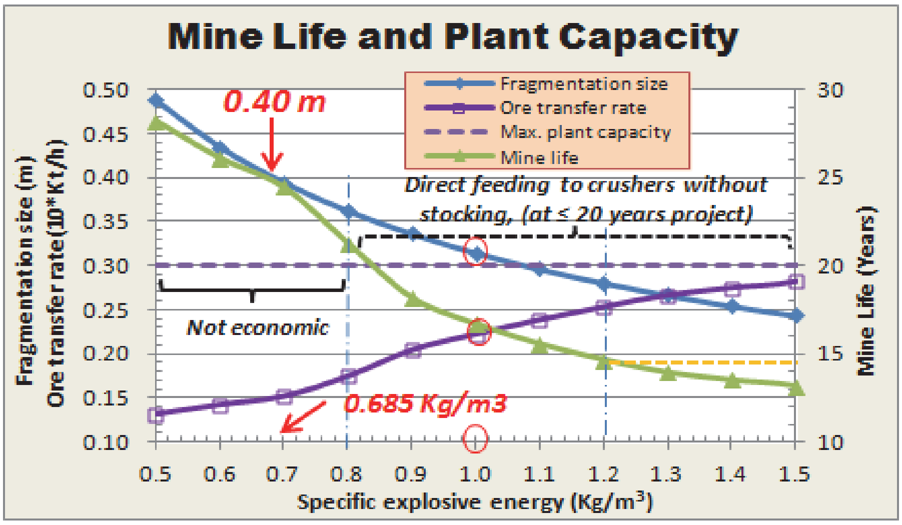


Fig. 46: Mine life range according to the ore delivery rates and the plant capacity.

The mine life range choice according to the previously mentioned constraints is illustrated in Figure 46. The figure shows the relation between the fragmentation size and the mine life indication across the span of the different investigated powder factors. The changing in the ore transfer rate of the ROM to the plant and its maximum allowable limits, which is the boundary of the maximum plant capacity, is shown also in the figure. It can be seen from the figure that the (State A), is located out of the previously indicated economic range, with a corresponding specific explosive energy equals 0.685 kg/m^3, (Table 10). Within the economic range, and until the end of the investigated span, the ore transfer rates are located under the limits of the maximum plant capacity (i.e. this allow the strategy of not stocking but direct feeding to the primary screens and crushers). The project life time of the economic range, which is located between 4.6 and 21.3 years, is seen also in the figure. The red circles indicate the chosen optimal fragmentation size (State B), which is according to both mining and processing activities, and will be detailed in the next sections.

5.4.2 Preliminary main results of the processing activities sub-model

Power and occupation limits for the different plant stages

Eleven runs are made within the chosen range of the specific explosive energy, which is between 0.5 and 1.5 kg/m^3 with a step of 0.1 kg/m^3. The collected total results are presented in Table 12. Processing activity results (crushing, coarse and fine grinding), which are presented in this table, are due to the data results from the previously chosen fleet strategy (Fleet A).

Figures 47 to 49 illustrate the required electric energies for the three mechanical size reduction operations across the span of the investigated specific explosive energy range and, hence, across the different delivered fragmentation size range. The figures are also showing the ore feeding rates for the three different types of the machinery lines.

In general, there is an obvious increasing in the energy requirements for the three stages by increasing of the applied explosive energy. This is because of the increasing in the amount of tonnage, which is received by the plant. As indicated in the previous section of the mining results, this higher amount of tonnage is due to the higher ROM transfer rate, which is accompanied with the decreased fragmentation size.

By the comparison between the three figures, it can be indicated that the tendency and the slope of the ore feeding rate trend for each stage is nearly the same, regardless of its component magnitudes. This is because that the effective factor for all of them is the same, which is the increasing in the ore delivery rates. However, the power consumption trends are clearly varies from the primary crushing to the fine grinding passing through the coarse grinding stage.

With the primary crushing stage, the power consumption slope is crawling gently upwards until reaching about 46 % of its starting value, however it becomes more steeply with the coarse grinding stage to reach more than 100 % of its starting value. With the fine grinding stage, the slope of the power consumption trend becomes more steeply upwards to reach about 110 % of its starting point.

The reason of the differences in these trend slopes is that the crushing process is more affected by the reduction in the feeding size, which enhancing the influence of the macro and micro-fissures, which are explained in (Appendix 1). This action makes as a softening for the primary crushing feed and reducing the bridging time through the jaw crushers.

Table 12: Processing results (primary crushing, coarse and fine grinding) for the specific explosive energy model runs.

Item/Run	**Unit**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Spec. explosive energy	kg/m^3	**0.50**	**0.60**	**0.70**	**0.80**	**0.90**	**1.00**	**1.10**	**1.20**	**1.30**	**1.40**	**1.50**
Energy required (pr. crushing)	kWh	611	623	632	709	766	826	843	892	900	897	893
Energy required (coarse grinding)	kWh	6671	7159	7601	8653	10052	10874	11603	12261	12846	13195	13502
Energy required (fine grinding)	kWh	22667	24431	26024	29786	34761	37747	40404	42796	44947	46263	47406
Prim. cr. bypass	t/h	93	119	142	193	257	310	359	404	451	488	517
Coarse gr. bypass	th	78	94	109	140	179	209	237	262	287	305	320
Fine gr. bypass	t/h	22	29	34	46	61	73	84	95	105	114	120
Prim. cr. specific energy	kWh/t	0.466	0.439	0.417	0.408	0.376	0.372	0.354	0.353	0.338	0.327	0.317
Coarse gr. specific energy	kWh/t	5.085	5.048	5.019	4.972	4.931	4.897	4.870	4.848	4.825	4.807	4.793
Fine gr. specific energy	kWh/t	17.279	17.226	17.184	17.115	17.054	17.001	16.957	16.921	16.883	16.852	16.829
Plant spec. energy	kWh/t	22.830	22.712	22.620	22.495	22.361	22.270	22.180	22.121	22.047	21.985	21.940
Plant prim. cr. utilization	%	45	48	51	57	66	71	75	79	82	84	85
Plant coarse gr. utilization	%	37	40	43	48	56	61	65	69	72	74	76
Plant fine gr. utilization	%	43	46	49	56	65	71	76	80	84	87	89
Specific CO_2 Emission (plant)	kg/t	2.420	2.408	2.398	2.384	2.370	2.361	2.351	2.345	2.337	2.330	2.326
Energy specific cost	\$/t	3.881	3.861	3.845	3.824	3.801	3.786	3.771	3.761	3.748	3.737	3.730
Processing specific cost	\$/t	3.881	3.861	3.845	3.824	3.801	3.786	3.771	3.761	3.748	3.737	3.730
Mining and processing cost	\$/t (milled)	8.411	8.424	8.404	8.314	8.266	8.250	8.250	8.266	8.358	8.455	8.547

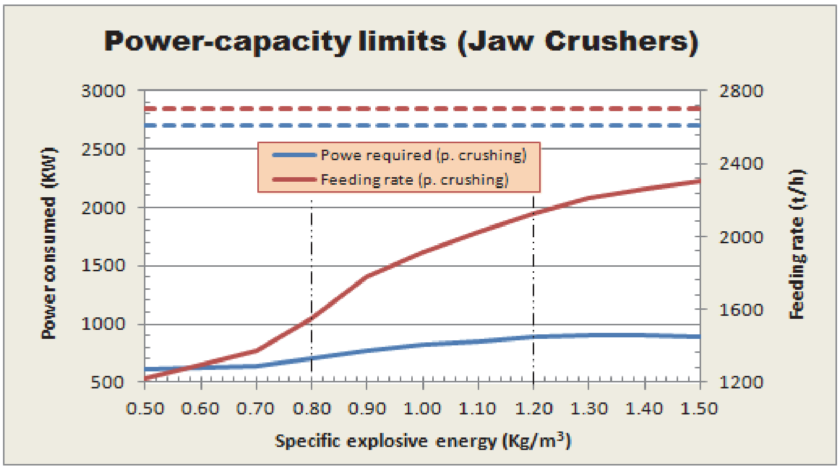


Fig. 47: Power and occupation limits for to the primary crushers.

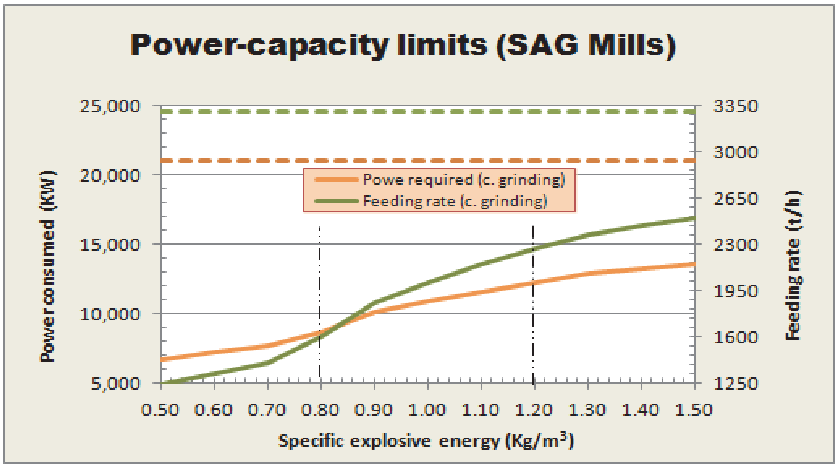


Fig. 48: Power and occupation limits for to the SAG mills.

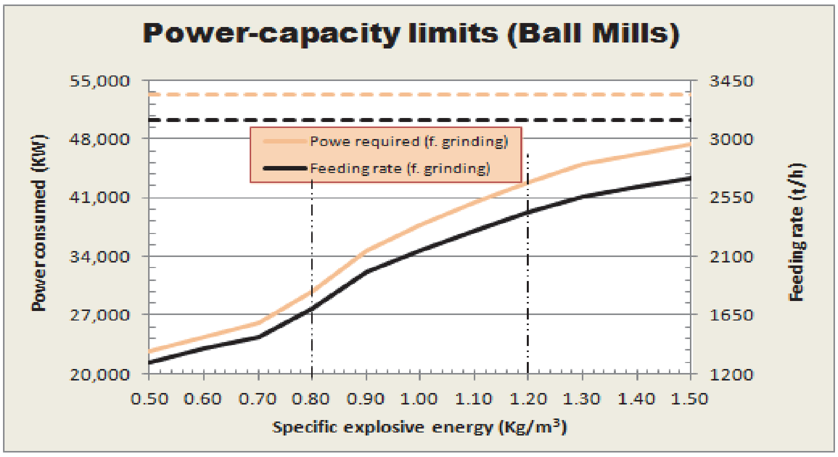


Fig. 49: Power and occupation limits for to the Ball mills.

However, this action is reduced gradually within the further size reduction stages to be approximately eliminated within the fine grinding, which has other effective factors causing its much higher magnitudes and steeper slope. These factors will be explained in the next section.

With regard to the power capacity and the occupation factor (loading degree) situations, each stage has its limits for the maximum power and feeding capacities. These limits are drawn for each stage figure by a dashed line with the same color of the corresponding solid line representing its operating magnitudes trend.

Within jaw crushers, the limiting factor is the loading degree, which reaches to more than 85 % of the total primary crushing available facilities, while the power limits play much less action with a maximum magnitude reaches not more than 33 % of its total availabilities. Within coarse grinding, the corresponding values are 64 % and 76 %, with more obvious rising for the two variables, especial with the power contribution of the available facilities.

Regarding the fine grinding stage, the corresponding values reach their maximum, which are 89 % and 86 %. This spots light on the critical situation for the fine grinding stage, which will be explained in the next section. It should be mentioned that, as the coarse grinding is only a transition stage between primary crushing and fine grinding, with a fixed feeding and product sizes and characteristics, just their magnitudes will be denoted, however the trends and behavior investigations will concentrated on the other two size reduction stages.

Comparison of the consumed energy requirements for the different stages

It is indicated in the upper paragraphs that the power consumptions for the three size reduction stages are rising, with different trends and slopes, by reducing the fragmentations size, mainly due to the increasing in the ore delivery rates to the plant. Nevertheless, the benefits of the size reduction of the delivered ROM will not be revealed unless the specific power consumption trend is investigated for each of them, which is illustrated in Figure 50.

From the figure, and also by referring to Table 12, it could be seen that the most energy consumption trend, which is affected and relatively reduced by decreasing of the plant feed fragmentation size, is that belongs to primary crushing, regardless of their absolute magnitudes. The reduction in the primary crushing power consumption reaches to about 67.6 % of the starting value, while within coarse grinding it is just about 94.1 %. The reduction within fine grinding is hardly felt, that is because it reaches not less than 97.3 % of the starting energy consumption value of the investigated span.

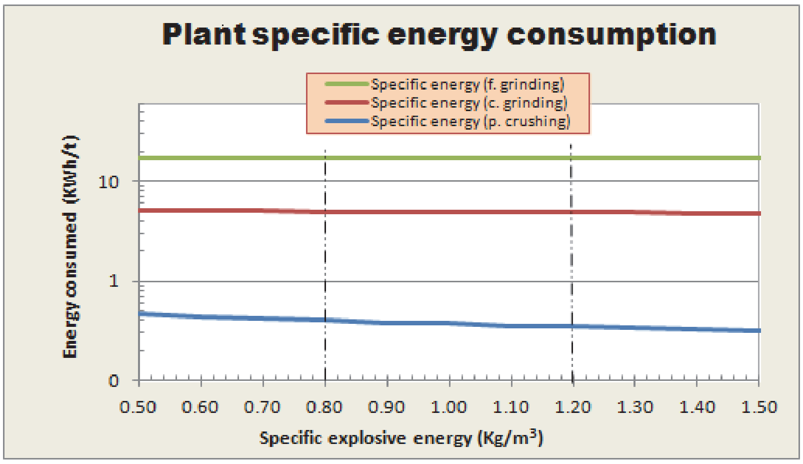


Fig. 50: Energy consumption for the different crushing and grinding stages.

The interpretation of the upper behavior can be found in Figure 51. The figure shows the specific energy consumption for the primary crushing and the fine grinding operations, accompanied with the amount of tonnage, which is by-passing each corresponding stage.

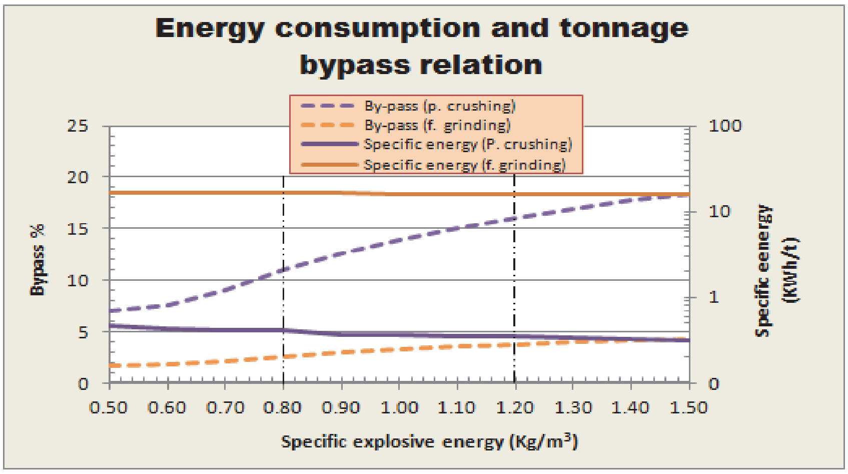


Fig. 51: Energy consumption and tonnage by-pass for crushing and fine grinding.

Regarding to the primary crushing, besides the previously mentioned reasons (section 5.4.1), which is belonging to the introduced fissures and causing the reduction in the required specific energy for crushing, the tonnage by-pass, due to the increasing of the fineness, plays an obvious role in this concern. The tonnage by-pass within this stage reaches to about 18.8 % of the delivered ROM at the end of the investigated span, which starts with just about 7 %. This is a considerable reason, which is belonging merely to the fragments size and causing the reduction in the consumed power.

However, with the fine grinding, the corresponding values are just about 4.4 % and 1.7 %, which act as a part of the reasons for the little unfelt reduction in the consumed power for fine grinding. The main other reason is belonging to the fact that the ore grain becomes relatively harder with decreasing of its size, as explained in Appendix 1. In addition to that the effect of the macro and micro-fissures, which resulted from the blasting operation, decreases gradually until almost being eliminated within this stage.

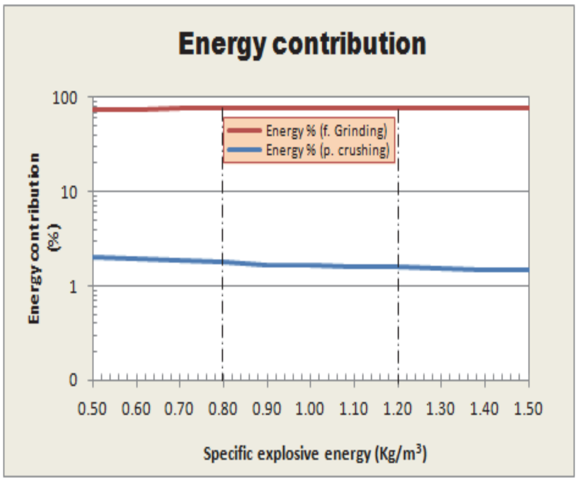


Fig. 52: Energy contributions comparison.

The contribution portion of the total plant electric energy requirements, for the two compared stages, is illustrated in Figure 52. It can be shown clearly the big difference between their requirements. While the energy consumption for the primary crushing stage contributes by just about (1.4 - 2.1) %, the corresponding value for the fine grinding stage is much more and ranges between 76 and 77 %.

The optimal fragmentation size and the optimization results

Figure 53 shows the summation for the mining and processing costs (drilling, blasting, loading, hauling, crushing and grinding) per the final milled ton. The points, which are corresponding to the specific explosive energies 1 and 1.1 kg/m^3, give approximately the same and the least values (8.250 \$/t$_{(milled)}$).

Thus, due to sustainability concerns and environmental considerations, such those belonging to the noise, vibration, blasting gas emissions,…etc, the explosive specific energy of the lower value (1 kg/m^3) will be chosen.

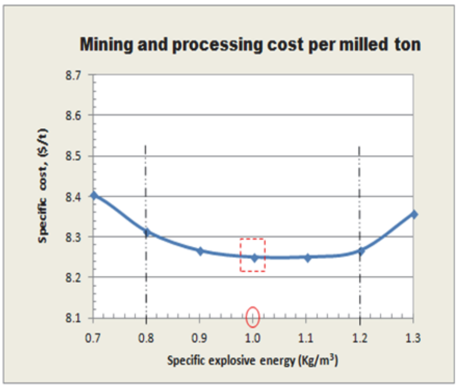


Fig. 53: The total costs per milled ton.

This chosen value is corresponding to a fragmentation size equals, 31 cm, and a mine life of 16.7 years, (Table 11). Thus, as we called the first comparable point for the reference mode installation the (State A), which assumed a current fragmentation size equals to 40 cm, we will call the chosen new fragments size as (State B).

Table 13, shows the main differences for the two cases, due to the fragmentation size improvement, from 40 to 31 cm.

Table 13: Comparison between the results for states A and B.

Item	Unit	State (A)	State (B)	Differ.
Loading and hauling	\$/T$_{(processed)}$	3.717	3.400	-8.53 %
Primary crushing (Energy)	\$/T$_{(processed)}$	0.072	0.063	-12.50 %
Coarse grinding (Energy)	\$/T$_{(processed)}$	0.855	0.832	-2.69 %
Fine grinding (Energy)	\$/T$_{(processed)}$	2.923	2.890	-1.13 %
Drilling and blasting	\$/T$_{(processed)}$	0.857	1.065	+24.27 %
Total costs	\$/T$_{(processed)}$	**8.424**	**8.250**	**-2.7 %**

By optimizing the drilling and blasting operations, by increasing their expenses to about 24 %, a reduction in the mean fragmentation size from 40 cm (state A) to 31 cm (state B) is resulted. This result improves the extraction and the transportation operations and shows good and reliable optimization (8.53 % reduction in its own specific cost). Regarding to the plant optimization, the primary crushing stage could be also be optimized (12.5 % energy cost reduction), while the other grinding stages showed no sensible optimization (just 2.69 % and 1.13 % reduction of the costs of the coarse and fine grinding stages, respectively). The overall reduction in the mining and processing costs will be 2.07 %, considering (Fleet A).

It should be mentioned that the main source for the cost optimization here is nearly the loading & hauling operation, which forms more than 31 % of the total project cost. Also, if we referred to (Fleet B), which is rejected from the former investigation of the mining activities (section 5.4.1), the reduction in the loading and hauling costs will increased to be 9.84 % instead of 8.53 %, and the total reduction in the mining and processing costs will be 2.7 % instead of 2.07 %. It should be mentioned also that, although the primary crushing shows good improvement in its specific cost, its real contribution and leverage in the total costs reduction is very low. That is because its share in the total project cost is less than 1 %, as it is illustrated in Figure 54.

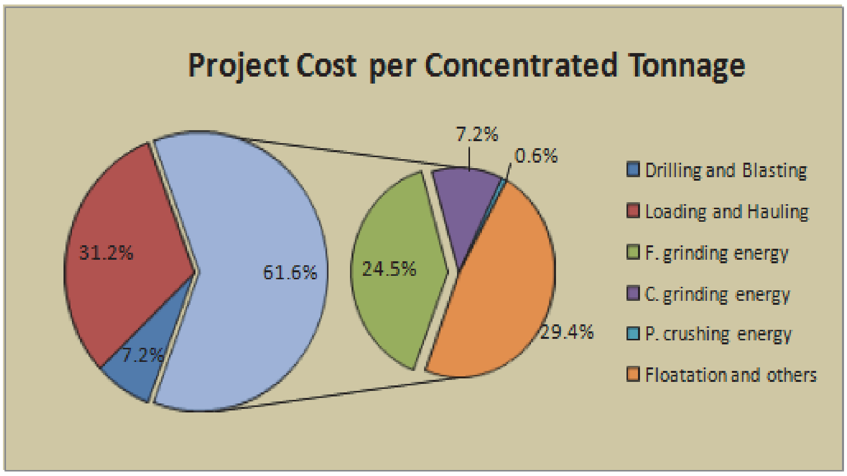


Fig. 54: Cost contributions for the different project operations.

The real area, which is calling for further investigation, within our previously mentioned scope of study, is the fine grinding stage. Despite this important stage forms about 25 % of the total expenses of the whole project, its optimization percent was just 1.13 %. This relatively low improvement declares that some other optimizations within the model should be done, in order to handle this problem and compensate for its predomination.

Sensitivity analysis for the dynamic model

As a fundamental test for the model robustness, a sensitivity analysis was done. This is made regarding to the final choice for the optimal mean blasting fragmentation size. As shown in Table 14, a changing in certain parameters is assumed. As it is pragmatic, increasing in the fuel price can be accompanied by increasing also in the electric energy price. Thus, different cases are assumed, which begin by decreasing in both prices by 10 %, then increasing of them by 10 % and 50 %. As shown from Table 17 and by the comparison with the original value for the optimal fragmentation size, the range of changing the magnitudes is (+ 1.6 % to -6.1%), which lies in the range of the accepted statistical error, which is ± 10 %.

Table 14: Sensitivity analysis results for the dynamic model.

Item	Unit	Value	- 10 %	+ 10 %	+ 50 %
Fuel (Diesel) price	$/L	1.25	1.12	1.37	2.5
Electric price	$/kWh	0.17	0.15	0.19	0.34
Optimal fragment size	cm	31	31.49	29.18	29.10
Statistical error	%	0 %	***+ 1.6 %***	***- 5.9 %***	***- 6.1 %***

5.4.3 Further model optimization requirements

As it is mentioned in the upper paragraphs, the fine grinding process is forming more than 75 % of the total energy consumed within the processing plant and about 25 % of the whole project costs. However, its optimization (the same with the coarse grinding stage) is not like the other stages, but it is obviously very low. An important reason for this is that the natural characteristics and parameters belonging to the ore type, hardness, micro-textures, liberation size, etc, are prevailing at this stage, not the other operational and technological factors.

As, this is only related to the physical prosperities of the mined ore deposit, it is intended to modify the model by providing an online selectivity in mining, in addition to design of different scenarios for the processing production lines. The mining and processing organization of the ore body is the core of the model optimization.

Every scenario will have a certain concept for how to process the three different ore types with a continuous, parallel, and separate technique. This can be, then, followed by a post-grinding mixing for the final ground products, in order to realize the maximum mineral recovery and the minimum fine grinding costs.

Therefore, new parameters with new functions will be introduced to the model, in order to realize this new concept. In the same time, the determined optimum fragmentation size (State B) as well as the determined ore transfer strategy (Fleet A) will be used within these further optimization.

6. The Model Optimization, Validation and Practical Applications

6.1 Model further optimization plan

Although the elementary results from the model (*Reference-mode*) led to good results in optimized planning for the mining and processing operations as one global process, as indicated in the previous chapter, the results are somewhat limited to the mining section. The plant results show improvement within primary crushing stage, while the other grinding stages showed no sensible optimization, which is calling for further investigation and optimization. The main reason for this is that the natural characteristics and parameters belonging to the ore type, hardness, micro-textures, liberation size,…etc, are prevailing at this stage, not the other operational and technological factors.

This is, fundamentally, related to the physical prosperities of the mined ore deposit, and therefore the optimization of the model will be basically by introducing suitable strategies with various scenarios for how to organize both of the mining and the processing operations, in order to overcome the problem of the ROM pluralism and heterogeneity.

As, shown in Figure 55, the flow chart for the model optimization plan is proposed by using of the previously chosen optimum fragmentation size (State B), with its corresponding project life time, as well as the determined ore transport strategy (Fleet A). Providing an online selectivity to the ore body extraction, according to a time-factored plan, in addition to the design of special organization for the processing production lines, is the core of the further model optimization.

Every scenario will have a certain concept for how to process the three different ore types with a continuous, parallel, and separate technique. This can be, then, followed by a post-grinding mixing for the final milled products, in order to realize the maximum mineral recovery and the minimum fine grinding costs.

Therefore, new parameters with new functions will be introduced to the model, which belong to organizing, financial and economic concepts, in order to realize this plan to differentiate and choose the optimal case, according to special indicators. The ore deposit, with its three ore types, will need to be, therefore, detailed according to their existence share within the mineralization area, the mining and refining cut-of-grades, and the grad-tonnage relationships and curves.

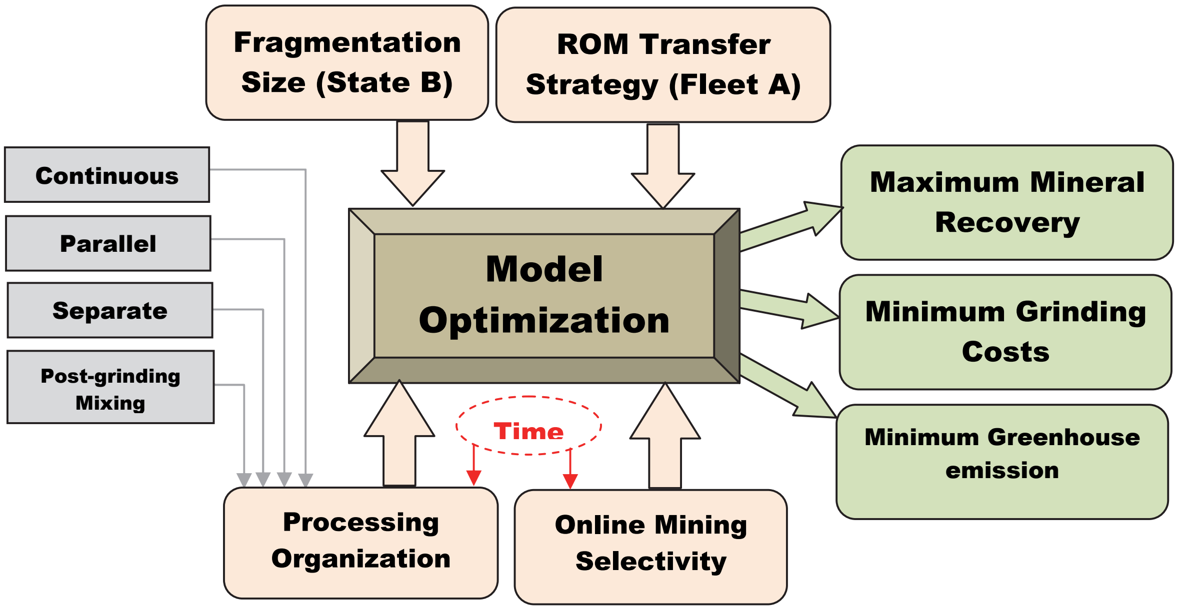


Fig. 55: Flow chart for the model optimization plan.

6.2 The ore deposit characteristics and details

6.2.1 Tonnage distribution and cut-off-grade for the ore deposit

The ore deposit existence in the mineralization area is detailed, in order to be appropriately mined and processed, according to its different constituents and physical properties. The *Grade-Tonnage* distribution and the mine and refining cut-off-grades for the whole deposit are shown in Figure 56 and Table 15.

The total ore reserves are 200 Mt, with a mine cut of grade of 0.35 % and an average ore grade 0.897 %. At 0.25 % there is more 68 Mt of low grade ore with an average grade of 0.3 %, which intended to be dumped alone.

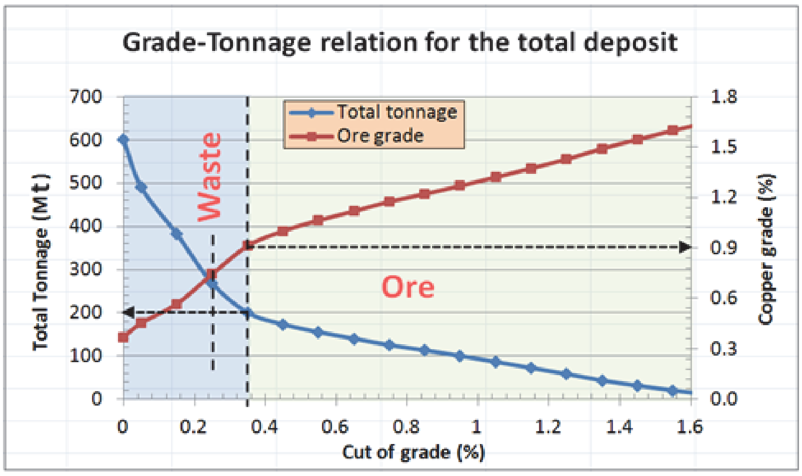


Fig. 56: Grade -Tonnage distribution according to the mine cut-off-grade.

Table 15: Grade -Tonnage distribution for the whole ore deposit area.

Cu grade (%) from - to		Grade M.P. (X)	Freq. (F), (Mt)	X*F	Cumu. tonnage, (Mt)	W. avg. Ore grade
<< 0.01 %		0	110	0	600.00	Overburden (600-200) = 400 Mt
0.01	***0.1***	0.05	108	5.40	490	
0.1	***0.2***	0.15	114	17.10	382	
0.2	***0.3***	**0.25**	**68**	17.00	268	
0.3	***0.4***	**0.35**	28.20	9.87	200.00	**0.897**
0.4	***0.5***	0.45	18.90	8.51	171.80	0.99
0.5	***0.6***	0.55	15.50	8.53	152.90	1.05
0.6	***0.7***	0.65	14.40	9.36	137.40	1.11
0.7	***0.8***	0.75	11.70	8.78	123.00	1.16
0.8	***0.9***	0.85	13.30	11.31	111.30	1.21
0.9	***1***	0.95	14.80	14.06	98.00	1.26
1	***1.1***	1.05	14.80	15.54	83.20	1.31
1.1	***1.2***	1.15	14.00	16.10	68.40	1.37
1.2	***1.3***	1.25	15.30	19.13	54.40	1.42
1.3	***1.4***	1.35	11.40	15.39	39.10	1.49
1.4	***1.5***	1.45	10.10	14.65	27.70	1.54
1.5	***1.6***	1.55	8.90	13.80	17.60	1.60
1.6	***1.7***	1.65	8.70	14.36	8.70	1.65

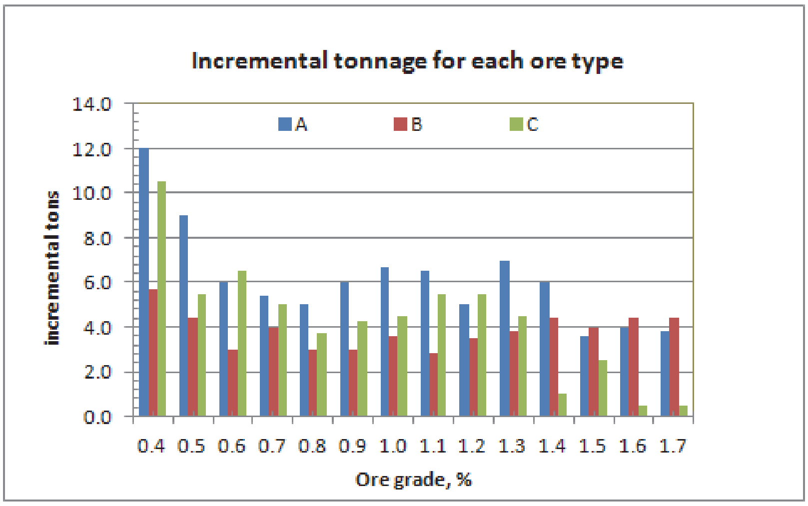


Fig. 57: Incremental tonnage for each ore type.

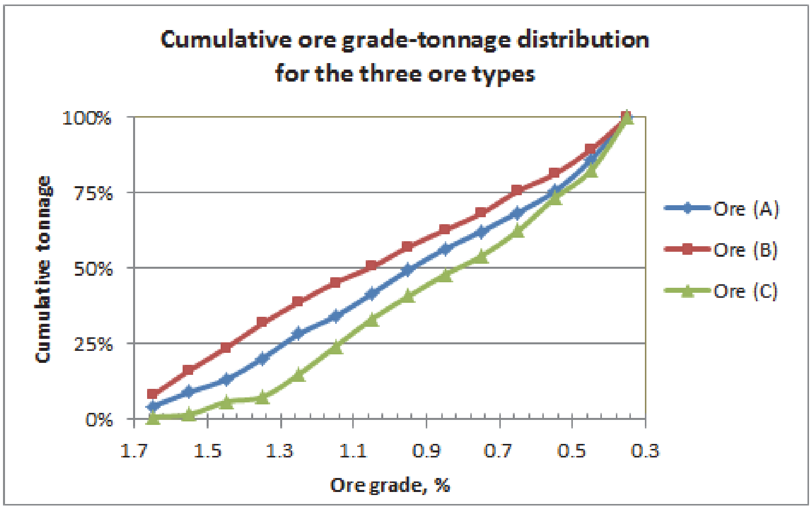


Fig. 58: Cumulative ore grade-tonnage distribution for the three ore types.

Grade -Tonnage distribution for each individual ore type within the ore deposit as well as the incremental and cumulative tonnage for each one are shown in Figure 57 & 58 and Table 16. It should be mentioned that the mine cut-off grade is the grade which can cover all the expenses of mining, processing and refining stages, while the refining cut-off grade is this grade which compensates just the expenses of the metal refining in the smelter.

In order to further understand the ore deposit characteristics, the next section will detail the liberation and microscopic grain size distribution for each contributed ore type.

Table 16: Grade -Tonnage distribution for each individual ore type within the ore deposit.

Grade M.P. (X)	Freq. (F), (Mt)	X*F	Cumu. tonnage (Mt)	W. avg. Ore grade-1	Freq. (F), (Mt)	X*F	Cumu. tonnage (Mt)	W. avg. Ore grade-2	Freq. (F), (Mt)	X*F	Cumu. tonnage (Mt)	W. avg. Ore grade-3
0.35	12.0	4.20	86.00	**0.90**	5.7	2.00	54.00	**1.00**	10.5	3.68	60.00	**0.80**
0.45	9.0	4.05	74.00	0.895	4.4	1.98	48.30	1.076	5.5	2.48	49.50	0.895
0.55	6.0	3.30	65.00	0.951	3.0	1.65	43.90	1.139	6.5	3.58	44.00	0.951
0.65	5.4	3.51	59.00	1.020	4.0	2.60	40.90	1.182	5.0	3.25	37.50	1.020
0.75	5.0	3.75	53.60	1.077	3.0	2.25	36.90	1.240	3.7	2.78	32.50	1.077
0.85	6.0	5.10	48.60	1.119	3.0	2.55	33.90	1.283	4.3	3.66	28.80	1.119
0.95	6.7	6.37	42.60	1.166	3.6	3.42	30.90	1.325	4.5	4.28	24.50	1.166
1.05	6.5	6.83	35.90	1.215	2.8	2.94	27.30	1.375	5.5	5.78	20.00	1.215
1.15	5.0	5.75	29.40	1.278	3.5	4.03	24.50	1.412	5.5	6.33	14.50	1.278
1.25	7.0	8.75	24.40	1.356	3.8	4.75	21.00	1.456	4.5	5.63	9.00	1.356
1.35	6.0	8.10	17.40	1.461	4.4	5.94	17.20	1.501	1.0	1.35	4.50	1.461
1.45	3.6	5.22	11.40	1.493	4.0	5.80	12.80	1.553	2.5	3.63	3.50	1.493
1.55	4.0	6.20	7.80	1.600	4.4	6.82	8.80	1.600	0.5	0.78	1.00	1.600
1.65	3.8	6.27	3.80	1.650	4.4	7.26	4.40	1.650	0.5	0.83	0.50	1.650
$\sum$	86	77.39	-	-	54	53.98	-	-	60	47.98	-	-

6.2.2 Liberation size and microscopic grain size distribution for the ore deposit

The mineral liberation grain size distribution for the three ore type contributions is illustrated in Figure 59. It could be seen from the figure how the grain size distribution for each individual ore type is overlapping the two others, with the shown statistical parameters on the upper right side.

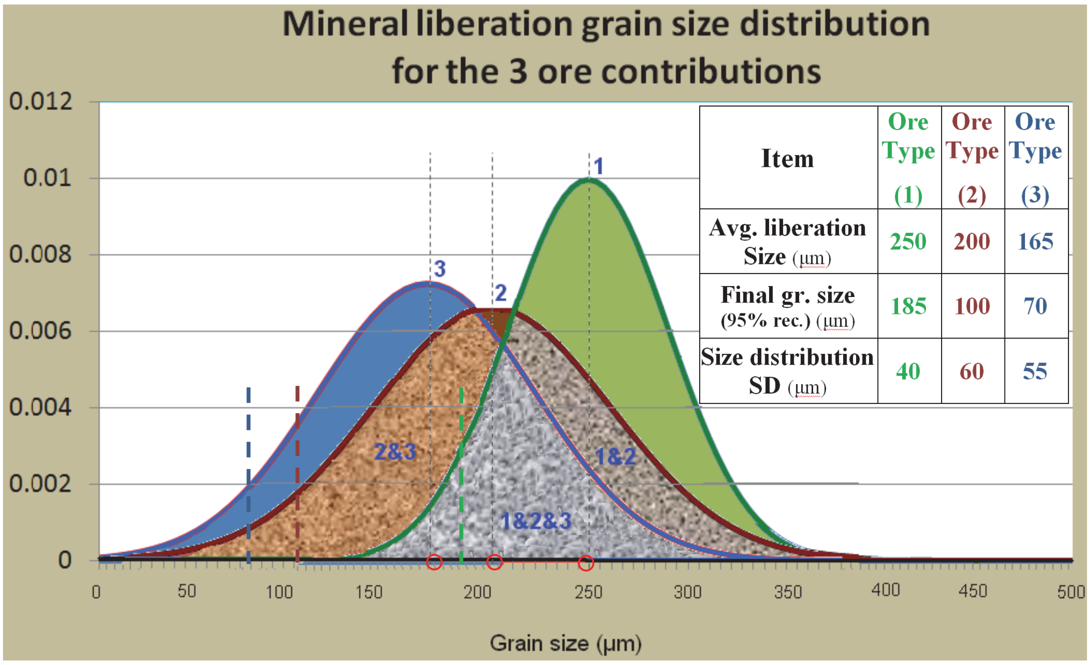


Fig. 59: Mineral liberation grain size distribution for the three ore type contributions.

The shown red circles indicate the average liberation size for each one, while the dashed lines indicate the grain size magnitude, which is representing the required final grinding size, in order to realize a mineral grains recovery of 95 %.

As each ore type has its own characteristic physical and chemical properties, and also as it has its own and different amount within the whole deposit (i.e. different exhaust time), the mining of the ore deposit should be based on special *time-based* selectivity plan, which will be described in the next section.

6.3 Mining selectivity and processing mixing scenarios

The main reason for the introducing of the selectivity in the ore extraction is the need for special designated plan for the ROM processing, in order to guarantee, not just the minimum overall specific costs, but also the integrated expression for the best mineral recovery, greenhouse emission and profitability.

The previous detected optimum fragmentation size with its corresponding project life and the chosen loading and hauling strategy will be fixed for the future working scenarios, which will depend principally on:

- The different ways for the ore types blending (pre- and post milling), and
- The special organization for the plant facilities.

Fourteen blending scenarios are chosen from the blending triangle, in order to investigate the dealing with the different plans for mining and processing of the three ore types according to their sharing in the ore fed to the plant allover the years of the project life.

6.3.1 Blending triangle design for choice of the annual mining contribution scenarios

As the ore deposit is consisting of different ore types, which are differing in their natural, chemical and mechanical properties, and as they are also different in their existence percent within the whole ore deposit, a time-factored mining and processing selectivity should be adopted to realize the followings:

- Guarantee of the real properties calculations belonging to the different operations,
- Guarantee stable tonnage feeding to the plant, and
- Guarantee better exploitation of the whole ore deposit by compensation for the low grade ore types.

A coding triangle, for the different blending scenarios, is designed and illustrated in Figure 60. Unlimited number of blending strategies for the three ore types A, B, and C, can be designated within the area of this triangle. The annual production sharing value (the coding value) for each blending code, which is located within the triangle, is illustrated in Table 17.

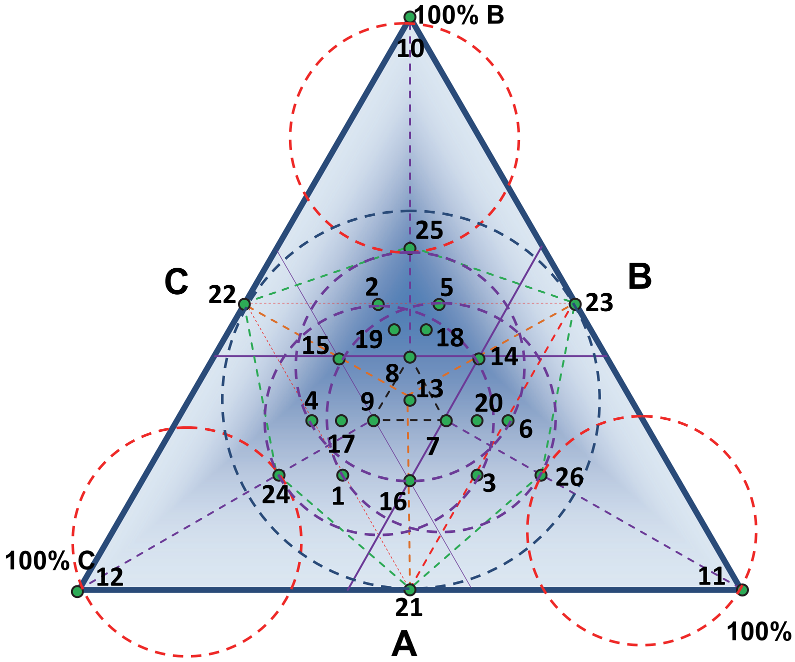


Fig. 60: The designed coding triangle for blending planning.

Table 17: The sharing value for each blending code;
The bold font is for the chosen scenarios.

Blending Code		1	2	3	4	5	6	7	8	9	10	11	12	13
Ore Type (share) (%)	A	**30**	**20**	50	**20**	**30**	50	40	**30**	**30**	0	100	0	**33**
	B	**20**	**50**	20	**30**	**50**	30	30	**40**	**30**	100	0	0	**33**
	C	**50**	**30**	30	**50**	**20**	20	30	**30**	**40**	0	0	100	**33**
Blending Code		**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**
Ore Type (share) (%)	A	40	**40**	20	**25**	**30**	**25**	45	50	0	50	**20**	**20**	60
	B	40	**20**	40	**30**	**45**	**45**	30	0	50	50	**20**	**60**	20
	C	20	**40**	40	**45**	**25**	**30**	25	50	50	0	**60**	**20**	20

Each point within the triangle indicates a fixed share for each individual ore type with respect to the whole annual feeding tonnage to the plant, regardless of its real existence within the ore deposit.

From the illustrated points, thirteen points are taken into account, which have the bold font within the table, plus one more point, which represents the natural existence for the ore-types. These fourteen points will be involved within the future various investigations. Each one represents a complete different scenario, (Table 18), for the annual production. Each scenario has its own criteria belongs to the different transition points such as the year at which each ore type will be terminated before the others, keeping the total annual production stable as much as possible …. and so on.

Table 18: The chosen mixing scenarios for the three ore types.

Mixing Scenario		1	2	3	4	5	6	7	8	9	10	11	12	13	14
Ore Type (share) (%)	A	30	20	20	30	30	30	33.3	40	25	30	25	20	20	43
	B	20	50	30	50	40	30	33.3	20	30	45	45	20	60	27
	C	50	30	50	20	30	40	33.3	40	45	25	30	60	20	30
Total (%) Production		*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*

6.3.2 Planed processing strategies according to the pre- and post-grinding mixing

The introducing of the selectivity in mining operations is normally should be followed by special organization to the plant operations, according to the annual different characteristics for the delivered ROM.

The special organization for the plant facilities is consisting of two strategies:

- Pre-grinding mixing (Processing strategy A), which is intending to send the total ore deposit for the total plant facility and deal with them as a bulk ROM, with weighted average values, (Fig. 61).

Post-grinding mixing (Processing strategy B), which is intending to send each individual ore type to a certain separated production line with its own characteristics and own set points, (Fig. 62).

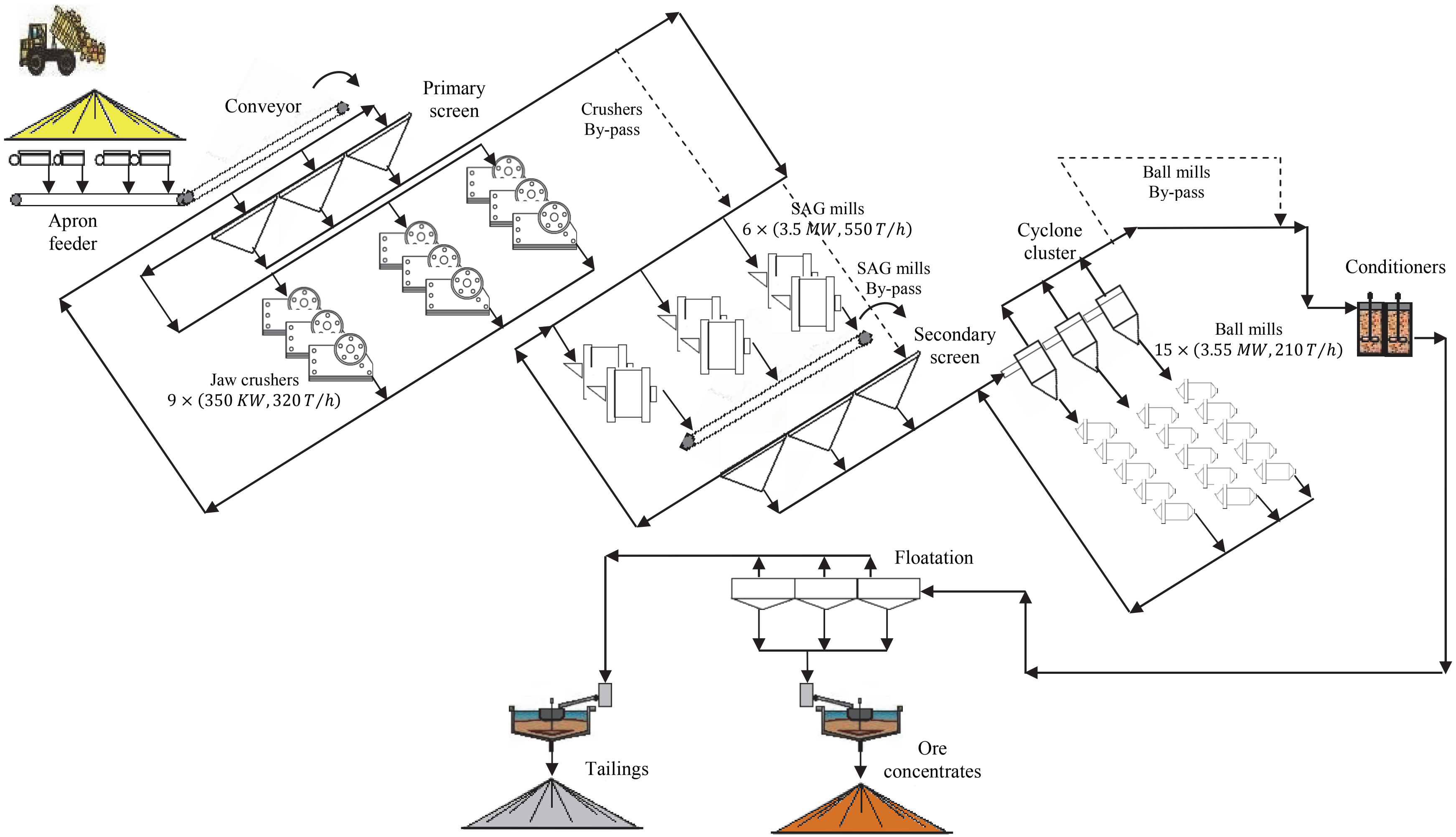


Fig. 61: Processing strategy A, the total ore deposit for the total plant facility (Pre-grinding mixing).

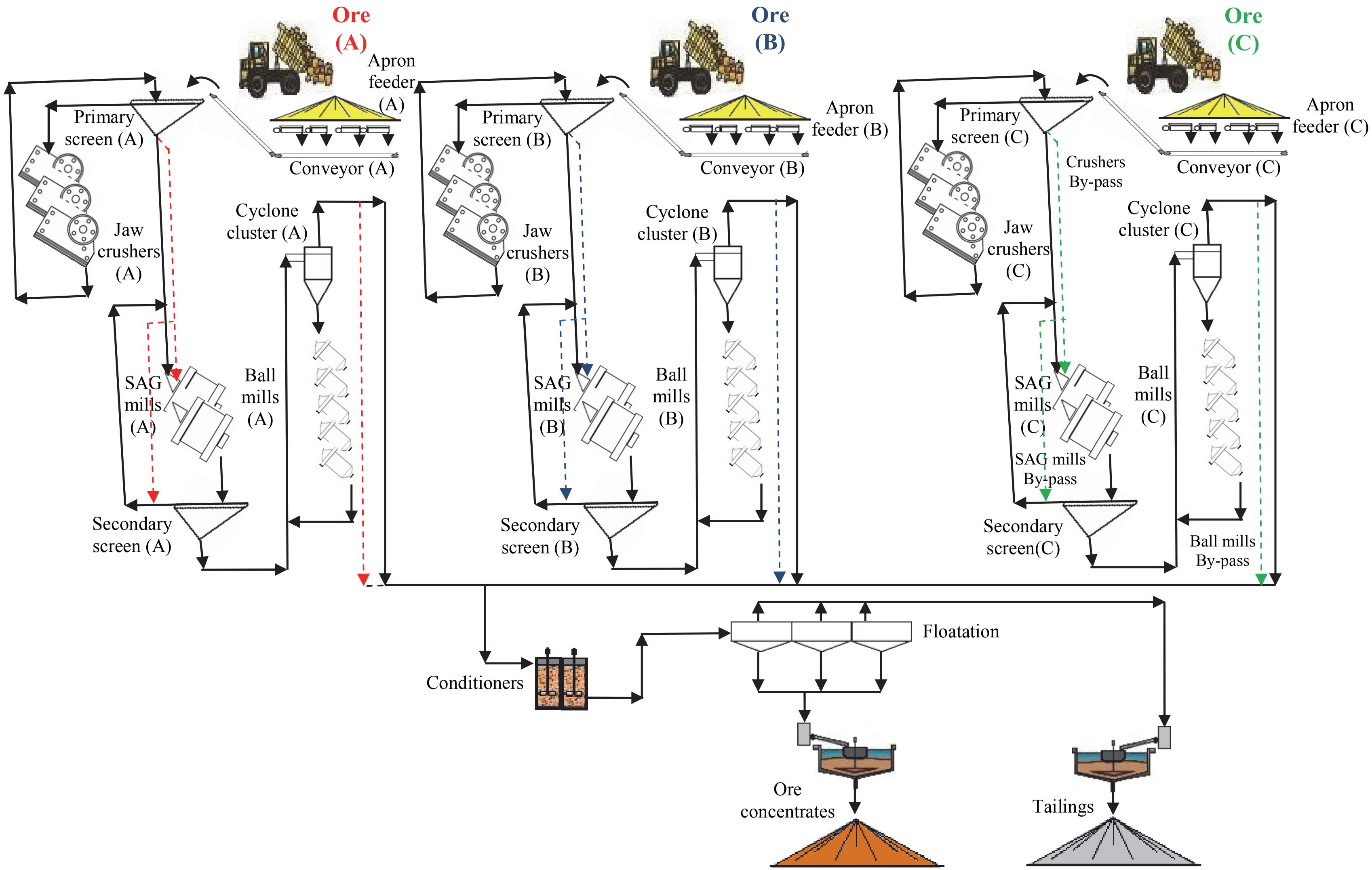


Fig. 62: Processing strategy B, the different ore types for separated processing lines (Post-grinding mixing).

It should be noticed that the set points belonging to each of the two strategies will be changing according to the different fourteen extraction scenarios. The set points and the calculated parameters will be changing also within each individual scenario due to any probable annual changing in the characteristics of its delivered ore. The plant facility is detailed previously in section (5.2).

6.4 An Excel calculation tool for preparing the new detailed inputs to the modified model

6.4.1 The need for new prepared and detailed inputs to the modified model

As the ore deposit is consisting of different ore types with different existence and properties within the mineralization area, in addition to the intended selective mining and processing due to different scenarios, a special calculation tool is constructed. This calculation tool is constructed in the form of an Excel file, in order to calculate the annual delivered ore tonnage and characteristics due to the three different ore types across the whole project life for the investigated blending scenarios.

The output results from the Excel calculation tool represent the main part of the inputs, which will be used in order to modify the Vensim dynamic model, which is intended to be concentrated mainly on the plant operations, due to the previous referred economical importance.

6.4.2 Description and benefits of the designed Excel calculation tool

The Excel file is consisting of one independent sheet for each blending scenario, in addition to a final sheet for the collected results, which is transferred to the Vensim model as a part of the modified inputs. Figure 63 shows a screenshot for a part of the Excel calculation tool for the mixing scenario No. 4.

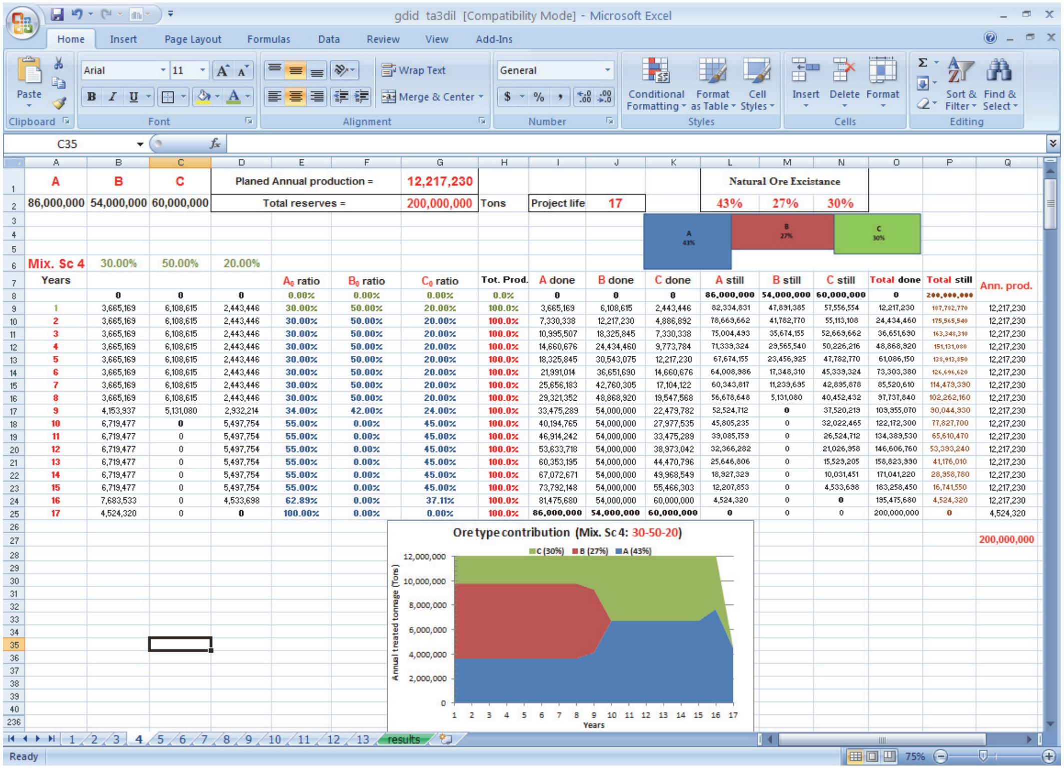


Fig. 63: Screenshot for a part of the Excel calculation tool for the mixing scenario No. 4.

Some design criteria were taken into account during the construction of the calculation tool such as:

- The total planed annual feeding should be the maximum, with respect to the chosen delivered fragmentation size, without exceeding the plant capacity.
- Across the project life, on termination or diminishing of one of the ore types, due to approaching its depletion, its share is divided equally between the remaining two others for the next year, unless one of them will be damped, terminated or diminished too.
- If the termination or diminishing of one of the ore types is accompanied also with diminishing of another one in the coming year, the remainder share of it will be divided for the other ore types according to the real rest tonnage of them within the ore deposit, with avoidance the risk of reducing of the required annual production.
- The final accumulative total product from each ore type at the end life of the project is equal to its real tonnage existence within the ore deposit.

- The calculation of the real annual production is depending on the previous cumulative production due to each individual ore type.

6.4.3 The main outputs of the Excel calculation tool

The output results of the calculation tool, which belong the ore types annual sharing tonnage across the project life, are illustrated in Figure 64.

Within each sub-figure, the annual tonnage production for each individual ore type for each one of the fourteen blending scenarios is illustrated. Some other points such as the year, at which the production of a certain ore type is increasing, decreasing or becoming zero, can be also observed from the sub-figures.

The calculated weighted averages for the other different output parameters, which are also part of the Excel calculation tool results, are illustrated in Table 19, for scenarios 1 to 3. The other tables belonging to the other 11 scenarios are illustrated in the Appendices, (Table Ap3-1).

The tables show the tonnage weighted averages for some important characteristic parameters of the delivered ROM, which will be used as an inputs part to the Vensim dynamic model optimization. Each value within the tables is characteristic to its different annual share contribution, according to their shown corresponding sub-figures.

6.4.4 The Excel calculation tool outputs as inputs to the modified Vensim model

Figure 65 illustrates a screenshot for the annual ore types share inputs, which are in fact the outputs from the Excel calculation tool.

Due to the different shares of the ore types across the mine life for each blending scenario, different delivered ore properties are handled, which are expected to have sensible impressions especially within the plant operations.

Fig. 64: Annual tonnage contribution and end point time of each ore type for the fourteen mixing scenarios (Output of the Excel calculation tool).

These time-factored shares are updating the annual magnitude for the main natural parameters within the model according to the values, which are detailed within the mentioned tables and sub-figures in section (6.4.3).

Table 19: Annual tonnage weighted averages of the main input parameters for the mixing scenarios (1-3) (Output of the Excel calculation tool).

Mix. Sc 1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	128.5	128.5	128.5	128.5	128.5	128.5	128.5	128.5	128.5	125.3	110.8	110.8	110.8	110.8	110.8	99.4	95.0
Work Index (FG)	17.70	17.70	17.70	17.70	17.70	17.70	17.70	17.70	17.70	17.30	15.45	15.45	15.45	15.45	15.45	15.13	15.00
WI (CG)	16.70	16.70	16.70	16.70	16.70	16.70	16.70	16.70	16.70	16.30	14.45	14.45	14.45	14.45	14.45	14.13	14.00
WI (PC)	15.70	15.70	15.70	15.70	15.70	15.70	15.70	15.70	15.70	15.30	13.45	13.45	13.45	13.45	13.45	13.13	13.00
Liberation size	198	198	198	198	198	198	198	198	198	203	228	228	228	228	228	244	250
Distribution st. Dev.	52	52	52	52	52	52	52	52	52	51	49	49	49	49	49	43	40
Metal content	0.87	0.87	0.87	0.87	0.87	0.87	0.87	0.87	0.87	0.88	0.95	0.95	0.95	0.95	0.95	0.91	0.90
Final size	111	111	111	111	111	111	111	111	111	117	147	147	147	147	147	174	185
Penetration rate	26.5	26.5	26.5	26.5	26.5	26.5	26.5	26.5	26.5	28.1	35.5	35.5	35.5	35.5	35.5	38.7	40.0
Discontinuity	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.4	0.9	0.9	0.9	0.9	0.9	0.8	0.7

Mix. Sc 2	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	128.4	128.4	128.4	128.4	128.4	128.4	128.4	128.4	127.7	124.2	124.2	124.2	122.6	95.0	95.0	95.0	95.0
Work Index (FG)	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.12	17.75	17.75	17.75	17.61	15.00	15.00	15.00	15.00
WI (CG)	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.12	16.75	16.75	16.75	16.61	14.00	14.00	14.00	14.00
WI (PC)	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.12	15.75	15.75	15.75	15.61	13.00	13.00	13.00	13.00
Liberation size	200	200	200	200	200	200	200	200	200	203	203	203	206	250	250	250	250
Distribution st. Dev.	55	55	55	55	55	55	55	55	53	48	48	48	48	40	40	40	40
Metal content	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.91	0.85	0.85	0.85	0.85	0.90	0.90	0.90	0.90
Final size	108	108	108	108	108	108	108	108	110	122	122	122	125	185	185	185	185
Penetration rate	28.1	28.1	28.1	28.1	28.1	28.1	28.1	28.1	28.0	27.4	27.4	27.4	28.0	40.0	40.0	40.0	40.0
Discontinuity	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.4	1.4	1.4	1.4	1.4	0.7	0.7	0.7	0.7

Mix. Sc 3	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	132.0	132.0	132.0	132.0	132.0	132.0	132.0	132.0	132.0	128.8	114.3	114.3	104.6	95.0	95.0	95.0	95.0
Work Index (FG)	17.80	17.80	17.80	17.80	17.80	17.80	17.80	17.80	17.80	17.40	15.55	15.55	15.28	15.00	15.00	15.00	15.00
WI (CG)	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.80	16.40	14.55	14.55	14.28	14.00	14.00	14.00	14.00
WI (PC)	15.80	15.80	15.80	15.80	15.80	15.80	15.80	15.80	15.80	15.40	13.55	13.55	13.28	13.00	13.00	13.00	13.00
Liberation size	193	193	193	193	193	193	193	193	193	198	223	223	236	250	250	250	250
Distribution st. Dev.	54	54	54	54	54	54	54	54	54	53	51	51	46	40	40	40	40
Metal content	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.89	0.96	0.96	0.93	0.90	0.90	0.90	0.90
Final size	102	102	102	102	102	102	102	102	102	108	138	138	162	185	185	185	185
Penetration rate	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	25.5	27.1	34.5	34.5	37.2	40.0	40.0	40.0	40.0
Discontinuity	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.4	1.0	1.0	0.8	0.7	0.7	0.7	0.7

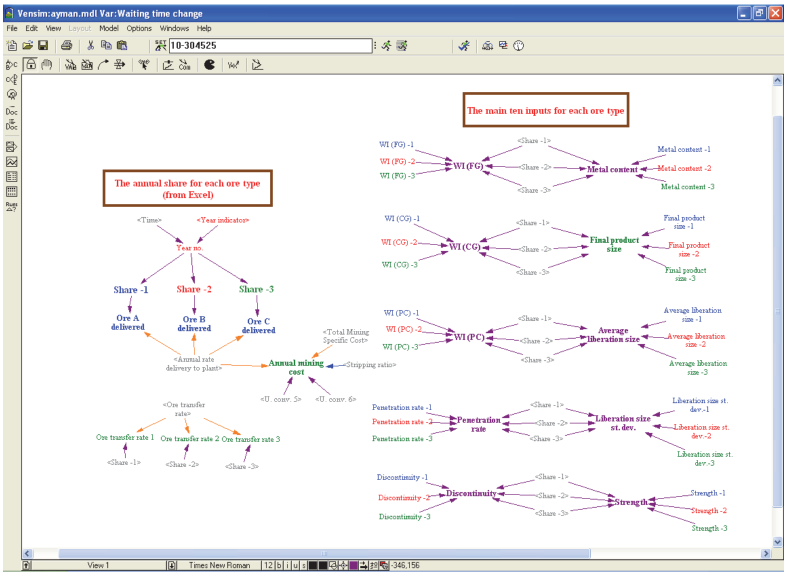


Fig. 65: Screenshot for the annual ore types share inputs.
(From the Excel calculation tool outputs)

6.5 The model optimization through the new added mathematical and functions

The degree of the metal recovery, due to the final stage of fine grinding, is the real chance or the probability that the metal grains are liberated and are available and ready to be recovered by the other concentration processes such as floatation. In order to introduce this function within the model, a lookup is installed to it, (Fig. 66).

This lookup is, in fact, the Z-factor of the probability distribution for the metal recovery calculations, which are summarized as:

$$\dot{M}_{rr} = M_{mill} \times m_r \times m_c \tag{89}$$

$$m_r = \prod \tag{90}$$

$$\prod = \frac{pX_{fg} - X_l}{\delta_X} \tag{91}$$

Where: M_{rr} metal recovery rate, (kg/h);

m_r milling recovery, (%); m_c metal content, (%);

X_l average liberation size, (µm);

δ_x liberation size standard deviation, (µm); and $\prod$ liberation probability.

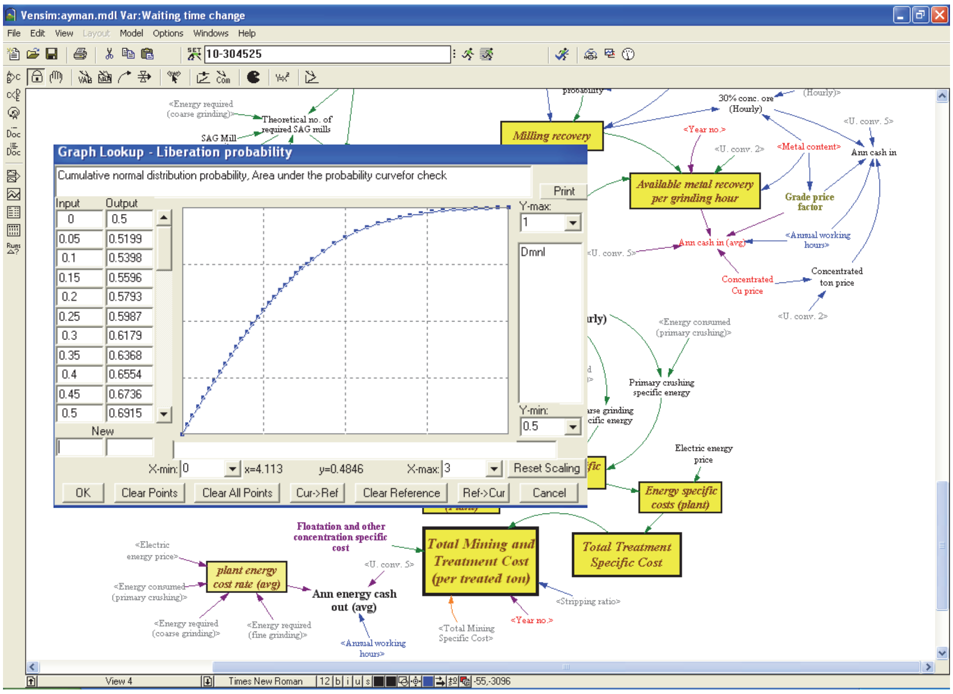


Fig. 66: Screenshot for the liberation probability and Z-factor function.

The liberation probability lookup, which is installed in the model, compares the calculated area under the microscopic size distribution curve with the typical Z-value in the Z-score table [117] and acts as a measure for the milling recovery.

The added financial functions and calculations can be summarized as:

$$\$_f = \$_{in} - \$_{cost} \tag{92}$$

$$\$_{cost} = SC_{M\&P} \times \dot{M}_{mill} \times n_d \times n_h \times 10^{-6} \tag{93}$$

$$\$_{in} = \$_{con.} \times \dot{M}_{rr} \times n_d \times n_h \times 10^{-6} \tag{94}$$

Where: $\$_f$ total annual cash flow, (M\$); $\$_{cost}$ annual costs, (M\$);

$\$_{in}$ annual income, (M\$); and $\$_{con.}$ concentrated metal ore price, (\$/t).

Present value calculations [6,139] are widely used in business and economics to provide a means to compare cash flows at different times for different projects or comparable project scenarios on a meaningful basis. Therefore, for a given scenario:

$$Max\ NPV = -\$_I + \left(\sum_{i=1}^{n_y} \frac{\$_{f,i}}{(1+r)^i}\right) \tag{95}$$

$$\$_{f,i} = 10^{-6} \times n_d \times n_h \left[\$_{conc.} \times \sum_{j=1}^{n_{ore}} \dot{M}_{rr_{j,i}} - \dot{M}_{mill_{j,i}} \times (SC_{energy,j} + SC_{f\&c} + SC_m)\right] \tag{96}$$

Where: i year number, $(1,2,\ldots..n_y)$; j ore type, $(1,2,\ldots..n_{ore})$;

n_{ore} ore type number in the ore deposit; r annual discount (interest) rate, (%);

Max NPV maximum (cumulative) net present value, (M\$);

$\$_I$ initial investments and other expenses before the production periods, (M\$);

$\$_{f,i}$ total discrete annual cash flow, (M\$);

$\dot{M}_{rr\,j,i}$ metal recovery rate for the ore type j in the year i, (kg/h);

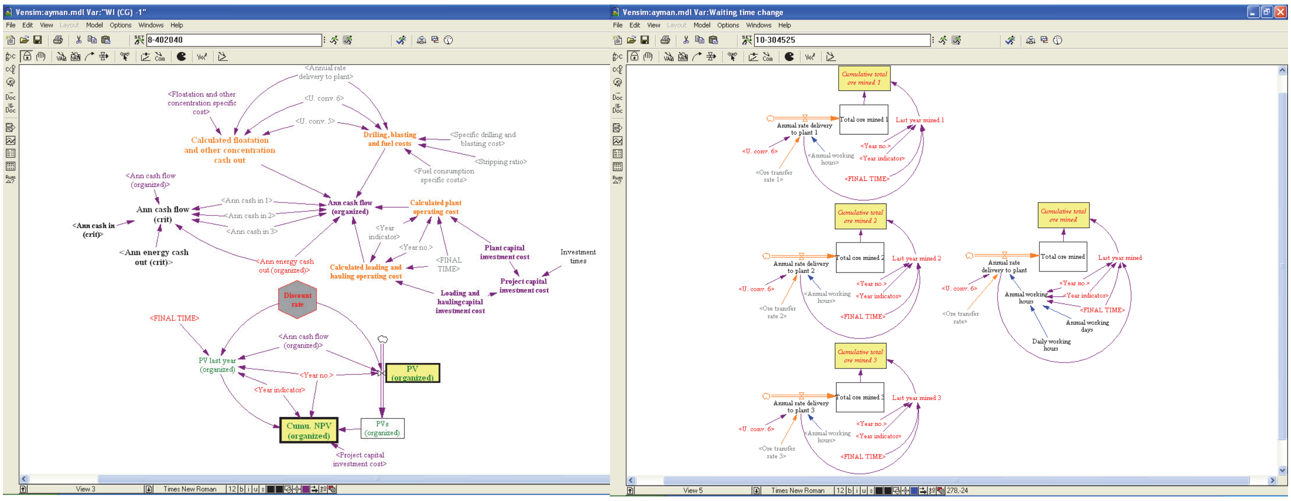
$\dot{M}_{mill\,j,i}$ total milled ore tonnage rate for the ore type j in the year i, (t/h); and

$SC_{energy,j}$ electric energy consumption specific cost for the ore type j, (\$/t).

In order to facilitate economical judgment of project alternatives, or different project scenarios, the internal rate of return (*IRR*), with the net present values calculations, are considered as key factors. The IRR [110] is the interest rate (%), which makes the Profitability Index (*PI*) equals to unity, or the cumulative net present value equals to zero, as:

$$\$_I \times (1+IRR)^{n_y+1} + \ \$_{f,1} \times (1+IRR)^{n_y} + \$_{f,2} \times (1+IRR)^{n_y-1} \cdots\cdots\cdots\cdots\cdots \$_{f,n_y} \times (1+IRR) = 0 \tag{97}$$

Figure 67 shows screenshots for two modified parts of the model, which belong to financial, economical, and time parameters, while Figure 68 shows a screenshot for a part of the modified *Crushing and Grinding* sub-model, belonging to ore-type 1.



(a) (b)

Fig. 67: Screenshots for two modified parts of the model.
(a) The financial and economical parameters.
(b) Time parameters and the ROM cumulative delivery.

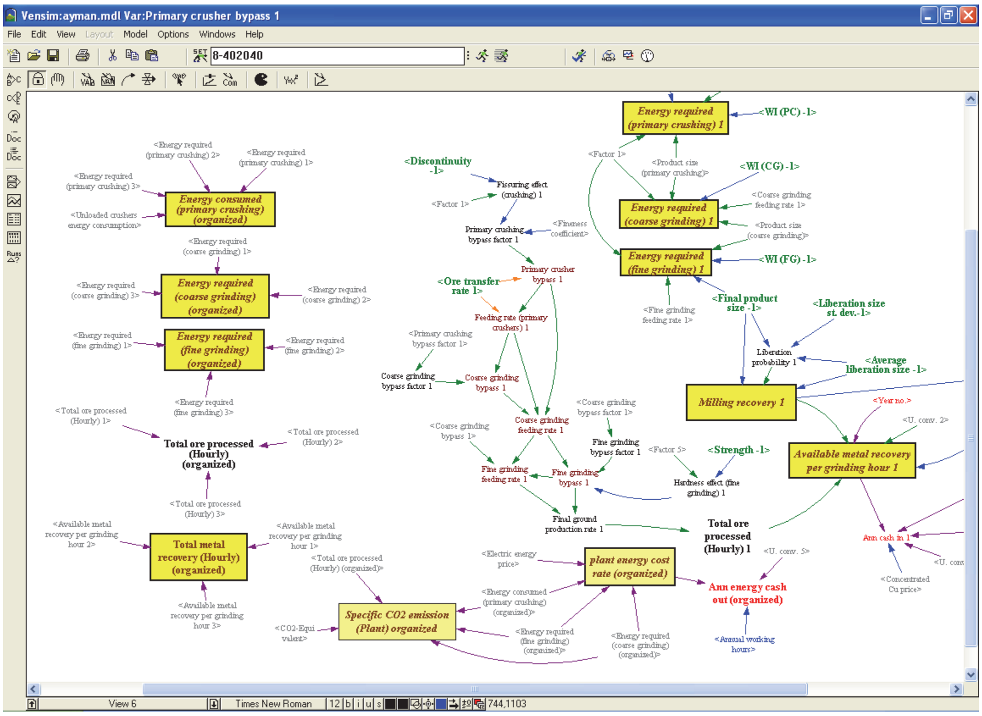


Fig. 68: Screenshot for a part of the modified *Crushing and Grinding* sub-model.
(Belongs to Ore type 1)

The Profitability Index [110] equation could be written as:

$$PI = \left\| \frac{\left(\sum_{i=1}^{n_y} \frac{\$_{f,i}}{(1+r)^i}\right)}{\$_I} \right\| \qquad (98)$$

The resultant different main characteristics and physical properties are dependent on the ore type natural share within the ore deposit, the chosen mixture ratios (scenarios on mining), and on the adopted processing strategy. Two main strategies are investigated: processing of the different mining scenarios after blending and processing them on individual or discrete production lines according to their original ore types.

Accordingly, the main physical properties for the ore deposit such as:

Strength σ, discontinuity (spacing) ε, metal content m_c, primary crushing work index $WI_{p.c}$, coarse grinding work index $WI_{c.g}$, fine grinding work index $WI_{f.g}$, penetration rate PR, average liberation size X_l, and the required final milled product size $pX_{p.c}$, will change as follows:

$$Z = f(i, j, \varsigma_j, l) \qquad (99)$$

Where: $i = 1,2,.....n_y$; $j = 1,2,....n_{ore}$; ς_j ore type natural sharing in the ore deposit, (%); and $l = 1,2,....n_{sc}$, where n_{sc} number of the investigated scenarios.

For any specific physical property Z, as the previously mentioned rock factors, it will be a function in the year indicator, the number of ore types within the ore deposits and their individual natural sharing in the ore deposit, and in the mixing scenario indicator.

The previous modified parameters, which include the outputs from the Excel file, is fed to the Vensim model as inputs, and the model is modified through the new links, lookups and functions in order to be able to process and update the whole previous and new equations.

As the previously chosen optimum fragmentation size with its corresponding project life and the chosen loading and hauling strategy are fixed, the improving in the mining activities, due to the further optimization, will be minor and the majority is concentrated within the plant activities.

Regarding the investigated mining and processing strategies (A) and (B) (section 6.3.2), which are based on different ways for the ore types handling and blending, each ore extraction method of the chosen fourteen scenarios will be an individual simulation run within the model. These simulation runs will be conducted for the lower data input and

processing methodologies.

The plant activities will be investigated through three different processing data methodologies. Two methodologies are under the mining and processing strategy (A) as follows:

- The *Average Values Method*, which applies the average data for the bulk ore-types characteristic parameters and the production lines set-points, abbreviated as (Avg); and
- The *Critical Ore-type Method*, which applies the data, which is belonging to the most critical ore type, such as the most stiff, tough and hard type, for the bulk ore-types, in order to obtain the maximum recovery through all the ore types, abbreviated as (Crit).

The third input data method is under the mining and processing strategy (B) and described as:

- The *Organized Method*, which applies, according to its own strategy, the corresponding characteristic parameters and production line set-points for each ore type individually and organizes the plant facilities, according to the ore-type, into parallel or series arranggements; and is abbreviated as (Org).

Table 20 shows the number of the main model introduced parameters, for each method.

Table 20: The number of the modified model introduced components.

Strategy	**Processing strategy (A)**		**Processing strategy (B)**
Input data method	Average Values *(Avg)*	Critical Ore-type *(Crit)*	Organized *(Org)*
General inputs	**5**	**5**	**5**
Specialized inputs	**10**	**10**	**30**
Intermediate parameters (without time factors)	**165**	**165**	**190**
Time and units stabilizing factors	**85**	**85**	**85**
Total components	**265**	**265**	**310**

It should be, also, mentioned that the number of the whole processed equations n_{eq} will be:

$$n_{eq} = n_y \times n_{ore} \times n_{sc} \tag{100}$$

6.6 The controlled model results and the comparable discussion of the processing strategies

6.6.1 General notifications for the model handling and the results presentation

The different characteristic scenarios data for the possibility of mining and processing of the ore deposit with its different ore types are coded and transferred from the Excel calculation tool to the modified Vensim model as fed lookups. The following General notifications are mentioned:

- The investigated scenarios are applied to the three previously mentioned data handling methods (arrangements) within the [*Controlled*] Vensim model, which are abbreviated as the (Avg), the (Crit) and the (Org) plant arrangements methods, as explained before.
- For each arrangement method, all the fourteen scenarios are investigated, yearly, across the whole project life (16.4 years).
- For each data arrangement method, selected tables, which include a certain effective output parameter for the overall scenarios, are represented here in this chapter, while the tables for the other main outputs are transferred to the appendices.
- For each data arrangement method, the most important output parameters for the corresponding (scenario 1) are represented here through two different tables, while the other tables for the other remainder scenarios are transferred to the appendices.
- Within the section of the comparison between the three data handling methods, beside representing and discussing the comparison figures, there will be referring to the corresponding tables numbers within the appendices, if required.

6.6.2 Results of the mining section of the model

As mentioned in the upper sections, the improvements in the mining activities, due to the further optimization, will be minor, compared to those belong to the plant activities, in which the optimization majority is concentrated.

That is mainly because the previously chosen optimum fragmentation size, with its corresponding project life, and the chosen loading and hauling strategy are fixed. Therefore, the drilling and blasting operations will be the main cause for any difference between the various scenarios, due to the different outputs of drilling energies and explosive amounts within each mining-type area.

Table 21 shows the drilling, blasting and fuel expenses for each extraction mixing scenario, while Table 22 shows the annual expenses of the special outputs, which will be fixed within the various processing methodologies. This will give the possibility to calculate all the mining, processing and concentration costs for the purpose of differentiation and making the preference according to the plant activities.

These expenses are normally excluding the crushing and grinding costs to facilitate tracing their effects, as they will be different, according to the year, the extraction scenario, and the plant arrangement method, which will be tabled in the next sections.

In Table 21, it can be observed that the difference between the extraction scenarios expenses are not so much, as the planed annual production tonnage and the ROM delivery rate to the plant is the same for the all. The main difference is introduced just due to the difference in the selected mining areas, from which the ore is extracted.

The main reason for representing the mining results here is that they will be added to the other processing expenses. This will give the actual image for the whole project and help in the judgment and better selection between the divergent results of the plant outputs due to the various scenarios and arrangement methods.

Table 21: Drilling, blasting and fuel costs for each mixing scenario, (M$).

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
*1	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.176	30.048
2	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.176	30.048
3	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.176	30.048
4	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.176	30.048
5	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.176	30.048
6	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.176	30.048
7	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.176	30.048
8	30.623	30.368	30.781	30.040	30.226	30.419	30.242	30.276	30.596	30.132	30.296	31.005	30.271	30.048
9	30.623	30.388	30.781	30.059	30.226	30.419	30.242	30.276	30.596	30.132	30.296	29.699	30.326	30.048
10	30.374	30.498	30.521	30.161	30.226	30.419	30.242	30.276	30.596	30.151	30.316	29.452	30.326	30.048
11	29.394	30.498	29.510	30.161	30.226	30.419	30.242	30.276	30.484	30.243	30.410	29.452	30.326	30.048
12	29.394	30.498	29.510	30.161	30.321	30.419	30.242	30.276	29.479	30.243	30.410	29.452	30.326	30.048
13	29.394	30.394	29.202	30.161	30.326	29.698	30.242	29.576	29.479	30.243	30.410	29.452	30.326	30.048
14	29.394	28.901	28.901	30.161	30.326	29.314	30.304	29.336	28.953	30.243	29.702	29.314	30.326	30.048
15	29.394	28.901	28.901	30.161	29.206	28.901	29.209	29.336	28.901	30.226	28.901	28.901	29.206	30.048
16	29.034	28.901	28.901	29.911	28.901	28.901	28.901	29.457	28.901	28.901	28.901	28.901	28.901	30.048
17	10.702	10.702	10.702	10.702	10.702	10.702	10.702	11.138	10.702	10.702	10.702	10.702	10.702	11.127
Total	**492.68**	**492.62**	**493.17**	**491.96**	**492.26**	**492.55**	**492.26**	**492.15**	**492.86**	**492.14**	**492.42**	**493.36**	**492.27**	**491.90**

*(Project capital investments, at year (0) = 203.2 M$)

Table 22: Annual expenses with excluding of crushing and grinding costs.

Item /Y	Extraction operating cost, M$	CO_2 emission (mine), kt	ROM delivery to plant, Mt	Plant operating cost, M$	Floatation and other conc. cost, M$
1	5.000	6.377	12.217	5.160	42.760
2	5.000	6.377	12.217	5.160	42.760
3	5.000	6.377	12.217	5.160	42.760
4	5.000	6.377	12.217	5.160	42.760
5	5.000	6.377	12.217	5.160	42.760
6	5.000	6.377	12.217	5.160	42.760
7	5.000	6.377	12.217	5.160	42.760
8	5.000	6.377	12.217	5.160	42.760
9	5.000	6.377	12.217	5.160	42.760
10	5.000	6.377	12.217	5.160	42.760
11	5.000	6.377	12.217	5.160	42.760
12	5.000	6.377	12.217	5.160	42.760
13	5.000	6.377	12.217	5.160	42.760
14	5.000	6.377	12.217	5.160	42.760
15	5.000	6.377	12.217	5.160	42.760
16	5.000	6.377	12.217	5.160	42.760
17	1.852	2.362	4.524	1.911	15.834
Total	**81.85**	**104.400**	**200.00**	**84.47**	**700.00**

6.6.3 Results of the processing section of the model

The *Average Values Method* main results and outputs

The plant arrangement *Average Values Method* (Avg) has the weighted average values of the different ore type's natural parameters as well as the plant facilities set-pints. Thus it utilizes both the tonnage shares percents and their average physical and chemical parameters.

(Tables 23-26) show the annual financial outputs for the total income, the total energy cost, the total project cash flow, and the project net present value, respectively, for the overall scenarios.

As a sample of the output results within this arrangement method, it can be observed that, while the scenario (Avg-8) has the highest income, hence the highest metal recovery, the scenario (Avg-4) was the best due to its higher cumulative present value. This is mainly due to the difference in the size reduction energy expenses between them accompanied by the earlier higher annual cash flow for the scenario (Avg-4).

Tables Ap4-1 to Ap4-8, within the Appendices, show the mass flow feeding rates and by-passes; the power consumption due to primary crushing, and fine grinding stages; the available metal recovery rates; and the CO_2 emissions, respectively, for the overall scenarios.

(Tables 27 & 28) show the collected important mass flow, required energies, metal recovery and financial outputs for (Avg-1). It should be mentioned here that the scenario (Avg-1) has the order number 12 between the other scenarios. This is referred to the delay in the metal recovery improvements across the project life, which is also accompanied by the early dealing with the harder parts of the ore deposit, which increases the energy expenses.

Scenario (Avg-1) is illustrated as a sample results, while the corresponding tables for the other scenarios are transferred to the Appendices, (Tables Ap5-1 to Ap5-13 and Ap6-1 to Ap6-13).

The *Critical Ore-type Method* main results and outputs

The plant arrangement *Critical Ore-type Method* (Crit) deals with the values of the most critical and harder ore type natural parameters as well as its corresponding plant facilities set-pints. Thus, it utilizes the extraction shares percents, from the Excel file, for each scenario, only for the tonnage calculations, while applies the physical and chemical parameters for just one ore type.

For the purpose of not repeating and concluding the discussions, all the tables belong to the (Crit) plant arrangement method, which are corresponding to those of the previously discussed (Avg) method, are transferred to (Appendices 7, 8, and 9).

The results discussion and interpretations, due to this plant arrangement method, will be included with the discussion of the comparison and method preference and choice, with the other two methods, in the texts of section (6.6.4).

The *Organized Method* main results and outputs

The 3rd processing data arrangement method, the *Organized Method* (Org), is belonging to the processing strategy (B) and the post-grinding mixing (sections 6.3.2 and 6.5). According to the (Org) method, the corresponding characteristic parameters and production line set-points for each ore type, individually, is applied.

Table 23: Annual income for each mixing scenario, (M$), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	237.590	225.887
2	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	237.590	225.887
3	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	237.590	225.887
4	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	237.590	225.887
5	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	237.590	225.887
6	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	237.590	225.887
7	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	237.590	225.887
8	202.926	225.356	201.569	238.115	226.176	214.418	222.273	214.660	208.118	232.102	225.855	190.003	215.833	225.887
9	202.926	221.025	201.569	233.345	226.176	214.418	222.273	214.660	208.118	232.102	225.855	248.216	203.018	225.887
10	213.485	197.877	212.764	207.842	226.176	214.418	222.273	214.660	208.118	227.439	221.403	259.714	203.018	225.887
11	258.389	197.877	260.777	207.842	226.176	214.418	222.273	214.660	213.037	205.489	205.236	259.714	203.018	225.887
12	258.389	197.877	260.777	207.842	204.196	214.418	222.273	214.660	265.968	205.489	205.236	259.714	203.018	225.887
13	258.389	201.013	252.574	207.842	203.018	247.761	222.273	245.782	265.968	205.489	205.236	259.714	203.018	225.887
14	258.389	239.671	239.671	207.842	203.018	256.237	208.045	256.889	242.137	205.489	220.448	256.237	203.018	225.887
15	258.389	239.671	239.671	207.842	232.603	239.671	232.542	256.889	239.671	205.984	239.671	239.671	232.603	225.887
16	245.809	239.671	239.671	214.897	239.671	239.671	239.671	259.818	239.671	239.671	239.671	239.671	239.671	225.887
17	88.750	88.750	88.750	88.750	88.750	88.750	88.750	96.352	88.750	88.750	88.750	88.750	88.750	83.646
Total	**3666.323**	**3626.280**	**3608.776**	**3688.964**	**3659.192**	**3645.106**	**3658.557**	**3691.650**	**3636.382**	**3672.718**	**3658.346**	**3631.425**	**3658.095**	**3697.838**

Table 24: Total energy annual cost for each mixing scenario, (M$), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	27.590	27.87
2	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	27.590	27.87
3	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	27.590	27.87
4	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	27.590	27.87
5	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	27.590	27.87
6	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	27.590	27.87
7	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	27.590	27.87
8	29.467	28.274	29.643	27.415	28.098	28.782	28.265	28.607	29.213	27.756	28.186	30.329	28.556	27.87
9	29.467	28.477	29.643	27.618	28.098	28.782	28.265	28.607	29.213	27.756	28.186	26.579	29.115	27.87
10	28.772	29.545	28.948	28.685	28.098	28.782	28.265	28.607	29.213	27.960	28.389	25.700	29.115	27.87
11	25.612	29.545	25.787	28.685	28.098	28.782	28.265	28.607	28.904	28.900	29.621	25.700	29.115	27.87
12	25.612	29.545	25.787	28.685	29.064	28.782	28.265	28.607	25.999	28.900	29.621	25.700	29.115	27.87
13	25.612	29.287	25.316	28.685	29.115	26.546	28.265	26.371	25.999	28.900	29.621	25.700	29.115	27.87
14	25.612	24.827	24.827	28.685	29.115	25.490	28.896	25.525	24.914	28.900	27.396	25.490	29.115	27.87
15	25.612	24.827	24.827	28.685	25.854	24.827	25.862	25.525	24.827	28.857	24.827	24.827	25.854	27.87
16	25.047	24.827	24.827	27.997	24.827	24.827	24.827	25.708	24.827	24.827	24.827	24.827	24.827	27.87
17	9.194	9.194	9.194	9.194	9.194	9.194	9.194	9.841	9.194	9.194	9.194	9.194	9.194	10.3203
Total	**456.276**	**456.267**	**456.300**	**456.235**	**456.249**	**456.272**	**456.222**	**456.248**	**456.791**	**456.244**	**457.172**	**456.344**	**456.250**	**456.2403**

Table 25: Total project annual cash flow for each mixing scenario, (M$), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	126.903	115.049
2	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	126.903	115.049
3	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	126.903	115.049
4	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	126.903	115.049
5	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	126.903	115.049
6	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	126.903	115.049
7	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	126.903	115.049
8	89.916	113.794	88.225	127.740	114.932	102.296	110.846	102.857	95.390	121.294	114.453	75.750	104.087	115.049
9	89.916	109.239	88.225	122.748	114.932	102.296	110.846	102.857	95.390	121.294	114.453	139.017	90.657	115.049
10	101.419	84.914	100.375	96.075	114.932	102.296	110.846	102.857	95.390	116.407	109.777	151.643	90.657	115.049
11	150.463	84.914	152.559	96.075	114.932	102.296	110.846	102.857	100.729	93.426	92.285	151.643	90.657	115.049
12	150.463	84.914	152.559	96.075	91.891	102.296	110.846	102.857	157.570	93.426	92.285	151.643	90.657	115.049
13	150.463	88.413	145.137	96.075	90.657	138.597	110.846	136.914	157.570	93.426	92.285	151.643	90.657	115.049
14	150.463	133.022	133.022	96.075	90.657	148.514	95.924	149.107	135.350	93.426	110.429	148.514	90.657	115.049
15	150.463	133.022	133.022	96.075	124.623	133.022	124.551	149.107	133.022	93.980	133.022	133.022	124.623	115.049
16	138.808	133.022	133.022	104.068	133.022	133.022	133.022	151.732	133.022	133.022	133.022	133.022	133.022	115.049
17	49.258	49.258	49.258	49.258	49.258	49.258	49.258	55.777	49.258	49.258	49.258	49.258	49.258	42.6026
Total	**1851.046**	**1811.068**	**1792.980**	**1874.446**	**1844.360**	**1829.965**	**1843.753**	**1876.921**	**1820.418**	**1858.019**	**1842.439**	**1815.401**	**1843.251**	**1883.387**

Table 26: Cumulative (net) present value for each mixing scenario, (M$), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	78.188	98.951	76.718	111.078	99.941	88.953	96.388	89.441	82.948	105.473	99.524	65.869	110.351	100.042
2	67.990	86.045	66.711	96.590	86.905	77.350	83.816	77.775	72.128	91.716	86.543	57.278	95.957	86.9935
3	59.121	74.822	58.009	83.991	75.569	67.261	72.883	67.630	62.720	79.753	75.255	49.807	83.441	75.6465
4	51.410	65.062	50.443	73.036	65.713	58.488	63.377	58.809	54.539	69.350	65.439	43.310	72.557	65.7796
5	44.704	56.576	43.864	63.510	57.141	50.859	55.110	51.138	47.426	60.305	56.903	37.661	63.093	57.1996
6	38.873	49.196	38.142	55.226	49.688	44.225	47.922	44.468	41.240	52.439	49.481	32.749	54.864	49.7388
7	33.803	42.779	33.167	48.022	43.207	38.457	41.671	38.668	35.861	45.599	43.027	28.477	47.708	43.2511
8	29.394	37.200	28.841	41.759	37.571	33.441	36.236	33.624	31.183	39.651	37.415	24.763	34.026	37.6097
9	25.560	31.053	25.079	34.893	32.671	29.079	31.509	29.239	27.116	34.479	32.535	39.517	25.770	32.7041
10	25.069	20.989	24.811	23.748	28.409	25.286	27.399	25.425	23.579	28.774	27.135	37.484	22.409	28.4383
11	32.341	18.252	32.792	20.651	24.704	21.988	23.826	22.109	21.651	20.081	19.836	32.595	19.486	24.729
12	28.123	15.871	28.514	17.957	17.175	19.120	20.718	19.225	29.451	17.462	17.249	28.343	16.944	21.5035
13	24.455	14.370	23.589	15.615	14.734	22.526	18.016	22.252	25.610	15.184	14.999	24.646	14.734	18.6987
14	21.265	18.800	18.800	13.578	12.812	20.989	13.557	21.073	19.129	13.204	15.607	20.989	12.812	16.2597
15	18.491	16.348	16.348	11.807	15.315	16.348	15.307	18.325	16.348	11.550	16.348	16.348	15.315	14.1389
16	14.834	14.215	14.215	11.121	14.215	14.215	14.215	16.215	14.215	14.215	14.215	14.215	14.215	12.2947
17	4.577	4.577	4.577	4.577	4.577	4.577	4.577	5.183	4.577	4.577	4.577	4.577	4.577	3.95888
Total	**394.997**	**461.905**	**381.419**	**523.959**	**477.149**	**429.961**	**463.325**	**437.399**	**406.520**	**500.612**	**472.887**	**355.427**	**505.062**	**485.787**

Table 27: Annual mass flow and power requirements for (mixing scenario 1), (*Avg-1*).

Item /Y	Feeding rate (P. C.), t/h	Feeding rate (C. G.), t/h	Feeding rate (F. G.), t/h	Primary crushers bypass, t/h	Coarse grinding bypass, t/h	Fine grinding bypass, t/h	Power required (P. C.), kW	Power required (C. G.), kW	Power required (F. G.), kW
1	1924	2038	2182	339	224	81	753	9688	21659
2	1924	2038	2182	339	224	81	753	9688	21659
3	1924	2038	2182	339	224	81	753	9688	21659
4	1924	2038	2182	339	224	81	753	9688	21659
5	1924	2038	2182	339	224	81	753	9688	21659
6	1924	2038	2182	339	224	81	753	9688	21659
7	1924	2038	2182	339	224	81	753	9688	21659
8	1924	2038	2182	339	224	81	753	9688	21659
9	1924	2038	2182	339	224	81	753	9688	21659
10	1921	2037	2181	342	225	81	739	9447	21156
11	1908	2031	2178	355	231	85	678	8354	18869
12	1908	2031	2178	355	231	85	678	8354	18869
13	1908	2031	2178	355	231	85	678	8354	18869
14	1908	2031	2178	355	231	85	678	8354	18869
15	1908	2031	2178	355	231	85	678	8354	18869
16	1903	2029	2176	360	233	86	667	8158	18460
17	1901	2028	2175	362	234	87	662	8082	18301

Table 28: Annual metal recovery and judgment financial data for (mixing scenario 1), (*Avg-1*).

Item /Y	Available metal recovery rate, klb/h	30% Concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	39.267	59	202.926	29.467	89.916	78.188	-125.012
2	39.267	59	202.926	29.467	89.916	67.990	-57.022
3	39.267	59	202.926	29.467	89.916	59.121	2.099
4	39.267	59	202.926	29.467	89.916	51.410	53.509
5	39.267	59	202.926	29.467	89.916	44.704	98.213
6	39.267	59	202.926	29.467	89.916	38.873	137.087
7	39.267	59	202.926	29.467	89.916	33.803	170.889
8	39.267	59	202.926	29.467	89.916	29.394	200.283
9	39.267	59	202.926	29.467	89.916	25.560	225.843
10	40.679	62	213.485	28.772	101.419	25.069	250.912
11	46.032	70	258.389	25.612	150.463	32.341	283.253
12	46.032	70	258.389	25.612	150.463	28.123	311.376
13	46.032	70	258.389	25.612	150.463	24.455	335.830
14	46.032	70	258.389	25.612	150.463	21.265	357.095
15	46.032	70	258.389	25.612	150.463	18.491	375.586
16	45.345	69	245.809	25.047	138.808	14.834	390.420
17	44.832	68	88.750	9.194	49.258	4.577	394.997

This means that the whole deposit is mined according to the annual tonnage shares (part of the Excel calculation tool output), while the processing inside the plant will be characteristic, synchronized and parallel. Each ore type will be handled separately and will have its own set-points and calculations.

As shown previously in Figure 64, the ore blending process will be just after all the size reduction stages and before the subsequent concentration processes (e.g. floatation).

(Tables 29-32) show the annual financial outputs for the total income, the total energy cost, the total project cash flow, and the project net present value, respectively, for the overall scenarios.

Within the Appendices, (Tables Ap10-1 to -6) show the mass flow by-passes (within fine grinding); the power consumption due to primary crushing, coarse grinding, and fine grinding stages; the available metal recovery rates; and the CO_2 emissions, respectively, for the overall scenarios.

(Tables 33 & 34) show the collected important data, such as the detailed power requirements, the detailed metal recovery, and the judgment financial outputs, for (Org-1).

Table 29: Annual income for each mixing scenario, (M$), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	251.043	228.4379
2	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	251.043	228.4379
3	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	251.043	228.4379
4	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	251.043	228.4379
5	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	251.043	228.4379
6	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	251.043	228.4379
7	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	251.043	228.4379
8	215.448	240.998	220.909	245.582	235.537	225.493	230.346	220.032	223.201	240.559	238.268	210.864	221.582	228.4379
9	215.448	234.796	220.909	239.380	235.537	225.493	230.346	220.032	223.201	240.559	238.268	246.885	204.527	228.4379
10	222.031	202.235	227.492	206.818	235.537	225.493	230.346	220.032	223.201	234.357	232.066	254.749	204.527	228.4379
11	252.018	202.235	257.480	206.818	235.537	225.493	230.346	220.032	226.127	205.672	205.426	254.749	204.527	228.4379
12	252.018	202.235	257.480	206.818	206.076	225.493	230.346	220.032	258.661	205.672	205.426	254.749	204.527	228.4379
13	252.018	203.610	242.734	206.818	204.527	246.703	230.346	241.243	258.661	205.672	205.426	254.749	204.527	228.4379
14	252.018	227.444	227.444	206.818	204.527	248.195	211.194	249.287	230.174	205.672	213.694	248.195	204.527	228.4379
15	252.018	227.444	227.444	206.818	221.943	227.444	221.898	249.287	227.444	205.901	227.444	227.444	221.943	228.4379
16	234.325	227.444	227.444	210.484	227.444	227.444	227.444	255.022	227.444	227.444	227.444	227.444	227.444	228.4379
17	84.222	84.222	84.222	84.222	84.222	84.222	84.222	104.444	84.222	84.222	84.222	84.222	84.222	84.5899
Total	3739.703	3739.652	3739.924	3739.650	3739.644	3739.922	3739.252	3739.663	3744.743	3739.648	3745.555	3740.101	3739.648	3739.596

Table 30: Total energy annual cost for each mixing scenario, (M$), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	31.663	30.9474
2	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	31.663	30.9474
3	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	31.663	30.9474
4	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	31.663	30.9474
5	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	31.663	30.9474
6	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	31.663	30.9474
7	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	31.663	30.9474
8	35.377	33.232	36.370	30.670	32.239	33.808	32.427	32.815	35.089	31.454	32.735	37.939	32.757	30.9474
9	35.377	33.462	36.370	30.900	32.239	33.808	32.427	32.815	35.089	31.454	32.735	27.863	33.390	30.9474
10	33.518	34.672	34.511	32.109	32.239	33.808	32.427	32.815	35.089	31.685	32.966	25.545	33.390	30.9474
11	25.048	34.672	26.042	32.109	32.239	33.808	32.427	32.815	34.263	32.750	34.362	25.545	33.390	30.9474
12	25.048	34.672	26.042	32.109	33.333	33.808	32.427	32.815	26.046	32.750	34.362	25.545	33.390	30.9474
13	25.048	33.903	23.360	32.109	33.390	27.817	32.427	26.824	26.046	32.750	34.362	25.545	33.390	30.9474
14	25.048	20.578	20.578	32.109	33.390	24.353	33.143	24.552	21.075	32.750	28.265	24.353	33.390	30.9474
15	25.048	20.578	20.578	32.109	23.653	20.578	23.679	24.552	20.578	32.622	20.578	20.578	23.653	30.9474
16	21.830	20.578	20.578	30.059	20.578	20.578	20.578	25.595	20.578	20.578	20.578	20.578	20.578	30.9474
17	7.620	7.620	7.620	7.620	7.620	7.620	7.620	11.298	7.620	7.620	7.620	7.620	7.620	11.4598
Total	506.600	506.591	506.641	506.590	506.590	506.640	506.576	506.594	507.096	506.591	507.713	506.686	506.591	506.6182

Table 31: Total project annual cash flow for each mixing scenario, (M$), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	136.284	114.522
2	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	136.284	114.522
3	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	136.284	114.522
4	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	136.284	114.522
5	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	136.284	114.522
6	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	136.284	114.522
7	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	136.284	114.522
8	96.528	124.478	100.838	131.952	120.153	108.345	114.756	104.021	104.596	126.053	122.316	89.000	105.635	114.522
9	96.528	118.025	100.838	125.500	120.153	108.345	114.756	104.021	104.596	126.053	122.316	136.402	87.890	114.522
10	105.219	84.145	109.539	91.627	120.153	108.345	114.756	104.021	104.596	119.601	115.863	146.832	87.890	114.522
11	144.656	84.145	149.007	91.627	120.153	108.345	114.756	104.021	108.460	89.759	87.733	146.832	87.890	114.522
12	144.656	84.145	149.007	91.627	89.503	108.345	114.756	104.021	150.216	89.759	87.733	146.832	87.890	114.522
13	144.656	86.393	137.253	91.627	87.890	136.268	114.756	131.922	150.216	89.759	87.733	146.832	87.890	114.522
14	144.656	125.044	125.044	91.627	87.890	141.609	94.826	142.480	127.226	89.759	102.805	141.609	87.890	114.522
15	144.656	125.044	125.044	91.627	116.164	125.044	116.090	142.480	125.044	90.133	125.044	125.044	116.164	114.522
16	130.541	125.044	125.044	97.595	125.044	125.044	125.044	147.050	125.044	125.044	125.044	125.044	125.044	114.522
17	46.304	46.304	46.304	46.304	46.304	46.304	46.304	62.412	46.304	46.304	46.304	46.304	46.304	42.4076
Total	**1874.099**	**1874.113**	**1873.784**	**1874.779**	**1874.477**	**1874.409**	**1874.092**	**1874.596**	**1878.470**	**1874.597**	**1879.103**	**1873.733**	**1874.473**	**1874.76**

Table 32: Cumulative (net) present value for each mixing scenario, (M$), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	83.938	108.242	87.685	114.741	104.481	94.213	99.788	90.454	90.953	109.611	106.362	77.392	118.507	99.5845
2	72.989	94.123	76.248	99.774	90.853	81.925	86.772	78.655	79.090	95.314	92.489	67.297	103.050	86.5952
3	63.469	81.847	66.303	86.760	79.002	71.239	75.454	68.396	68.774	82.882	80.425	58.519	89.609	75.3002
4	55.190	71.171	57.655	75.444	68.698	61.947	65.612	59.475	59.803	72.071	69.935	50.886	77.921	65.4784
5	47.992	61.888	50.134	65.603	59.737	53.867	57.054	51.717	52.003	62.671	60.813	44.249	67.757	56.9378
6	41.732	53.815	43.595	57.046	51.945	46.841	49.612	44.971	45.220	54.496	52.881	38.477	58.919	49.5111
7	36.289	46.796	37.909	49.606	45.170	40.731	43.141	39.106	39.322	47.388	45.983	33.459	51.234	43.0531
8	31.555	40.692	32.964	43.135	39.278	35.418	37.514	34.005	34.193	41.207	39.985	29.094	34.532	37.4375
9	27.439	33.550	28.665	35.675	34.155	30.799	32.621	29.569	29.733	35.832	34.770	38.774	24.984	32.5544
10	26.009	20.799	27.077	22.649	29.700	26.781	28.366	25.713	25.855	29.564	28.640	36.295	21.725	28.3081
11	31.093	18.086	32.028	19.695	25.826	23.288	24.666	22.359	23.313	19.293	18.858	31.561	18.891	24.6158
12	27.037	15.727	27.851	17.126	16.729	20.251	21.449	19.442	28.076	16.777	16.398	27.444	16.427	21.405
13	23.511	14.041	22.308	14.892	14.285	22.147	18.651	21.441	24.414	14.588	14.259	23.864	14.285	18.6131
14	20.444	17.672	17.672	12.950	12.421	20.013	13.402	20.137	17.981	12.686	14.529	20.013	12.421	16.1853
15	17.778	15.367	15.367	11.261	14.276	15.367	14.267	17.510	15.367	11.077	15.367	15.367	14.276	14.0741
16	13.950	13.363	13.363	10.430	13.363	13.363	13.363	15.714	13.363	13.363	13.363	13.363	13.363	12.2384
17	4.303	4.303	4.303	4.303	4.303	4.303	4.303	5.800	4.303	4.303	4.303	4.303	4.303	3.94076
Total	**421.517**	**508.284**	**437.926**	**537.889**	**501.021**	**459.293**	**482.833**	**441.263**	**448.560**	**519.923**	**506.158**	**407.157**	**539.003**	**482.6328**

Table 33: Annual detailed power requirements (kW) for (mixing scenario 1), (*Org-1*).

Item /Y	Power required (PC)-1	Power required (PC)-2	Power required (PC)-3	Total power (PC)	Power required (CG)-1	Power required (CG)-2	Power required (CG)-3	Total power (CG)	Power required (FG)-1	Power required (FG)-2	Power required (FG)-3	Total power (FG)
1	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
2	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
3	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
4	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
5	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
6	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
7	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
8	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
9	122	88	287	498	2424	1737	5531	9693	4102	4771	19218	28090
10	140	108	236	484	2788	2128	4536	9452	4717	5844	15759	26320
11	224	199	0	422	4445	3909	0	8354	7520	10734	0	18254
12	224	199	0	422	4445	3909	0	8354	7520	10734	0	18254
13	224	199	0	422	4445	3909	0	8354	7520	10734	0	18254
14	224	199	0	422	4445	3909	0	8354	7520	10734	0	18254
15	224	199	0	422	4445	3909	0	8354	7520	10734	0	18254
16	355	56	0	411	7063	1095	0	8158	11950	3005	0	14955
17	407	0	0	407	8082	0	0	8082	13673	0	0	13673

Table 34: Annual metal recovery and judgment financial data for (mixing scenario 1), (*Org-1*).

Item /Y	Metal recovery rate (1), klb/h	Metal recovery rate (2), klb/h	Metal recovery rate (3), klb/h	Available metal recovery rate, klb/h	Total 30% concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	83.938	-119.262
2	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	72.989	-46.273
3	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	63.469	17.196
4	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	55.190	72.386
5	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	47.992	120.378
6	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	41.732	162.110
7	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	36.289	198.398
8	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	31.555	229.953
9	12.763	9.497	19.109	41.369	63	215.448	35.377	96.528	27.439	257.393
10	14.678	11.634	15.669	41.981	63	222.031	33.518	105.219	26.009	283.401
11	23.400	21.368	0.000	44.767	68	252.018	25.048	144.656	31.093	314.494
12	23.400	21.368	0.000	44.767	68	252.018	25.048	144.656	27.037	341.532
13	23.400	21.368	0.000	44.767	68	252.018	25.048	144.656	23.511	365.042
14	23.400	21.368	0.000	44.767	68	252.018	25.048	144.656	20.444	385.486
15	23.400	21.368	0.000	44.767	68	252.018	25.048	144.656	17.778	403.264
16	37.184	5.983	0.000	43.167	65	234.325	21.830	130.541	13.950	417.214
17	42.545	0.000	0.000	42.545	64	84.222	7.620	46.304	4.303	421.517

Scenario (Org-1) is illustrated as a sample result, while the corresponding tables for the other scenarios are transferred to the Appendices, (Tables Ap11-1 to Ap11-13 and Ap12-1 to Ap12-13).

The results discussion and interpretations, due to this plant arrangement method, will be included with the discussion of the comparison and method preference and choice, with the other previous two methods, in the texts of the next section.

6.6.4 Comparison between the three data processing and arrangement methodologies

Recovery results comparison for the arrangement methodologies

The metal recovery results are the real indicator for the liberation degree due to the various size reduction mechanisms, especially within the fine grinding stage. The results of the metal recovery reflect directly the amount of the expected concentrated tonnage due to the subsequent concentrating technologies, such as floatation and, hence, the amount of the annual income due to shipping to the smelter (for refining).

In order to be deeper investigated, the recovery is estimated in the model as two functionalities, which are proportional and typical. The first is more zooming in the liberation degree due to fine grinding and is estimating the expected liberated copper amount due to one grinding hour. The other is more zooming in the financial concepts and estimating the expected amount of the concentrated tonnage to 30 % metal content after the floatation process and reflects the real income due shipping to the smelter.

The recovery rate results due to the (Avg), the (Crit), and the (Org) arrangement methods are drawn together and illustrated in Figure 69, which could be reviewed in Table Ap4-7, Table Ap7-7, and Table Ap10-5, respectively. The yearly detailed data for each individual scenario is also presented in the even numbered tables in (Appendices 6, 9, and 12), respectively.

It can be shown clearly from the figure that the (Crit) method gives the heights metal recovery, which is just about 0.6 % more than t hat of the (Org) method, while reaches to about 3.8 % more than the (Avg) method. It can be noticed that both of the (Crit) and the (Org) methods gives almost a straight line trend, which means fixed metal recovery production, while with the (Avg) method it is completely different.

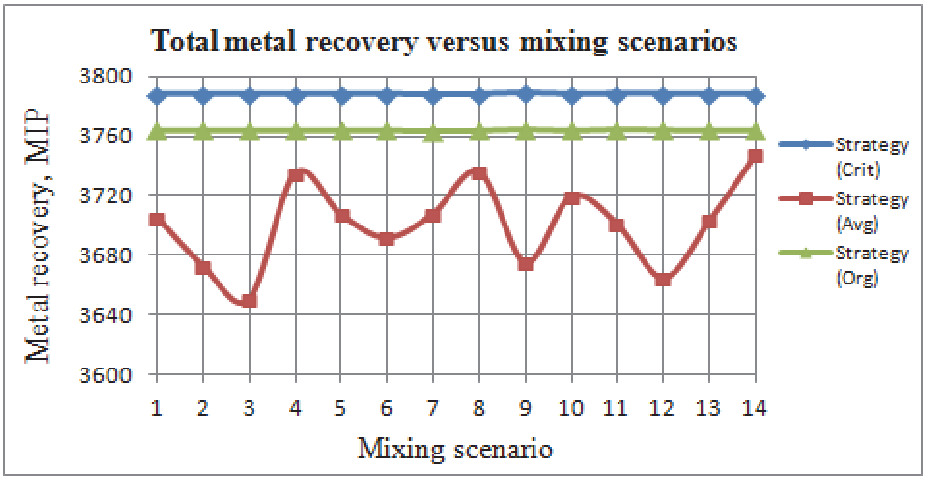


Fig. 69: Metal recovery versus mixing scenarios for the three processing strategies.

With regard to the (Avg) method, the metal recovery path is taking the figure, which is resulting from the reaction between the different annual ROM delivery to the plant, due to both the ore-type tonnage and the used average values of the different annual set-points of the plant facilities.

It should be mentioned also that the main reason for yielding of the fixed and straight line trend for the metal recovery, with the (Crit) and (Org) methods, is that both of them utilize a fixed unique final product set-pint for all scenarios across the whole project life.

Therefore, despite of the introduced different tonnage prosperities delivered to the plant facilities, the recovery output is nearly fixed.

Financial and economic results comparison for the arrangement methodologies

As mentioned in section 6.5, the present value calculations provide a means to compare cash flows at different times for different projects or comparable scenarios on a meaningful basis, in order to be better able to judge their feasibility.

The net present value for the whole suggested scenarios for the (Avg), the (Crit), and the (Org) arrangement methods, which could be reviewed in Table 26, Table Ap7-3, and Table 32, respectively, are drawn together and illustrated in Figure 70. The yearly detailed data for each individual scenario is also presented in the even numbered tables in (Appendices 6, 9, and 12), respectively.

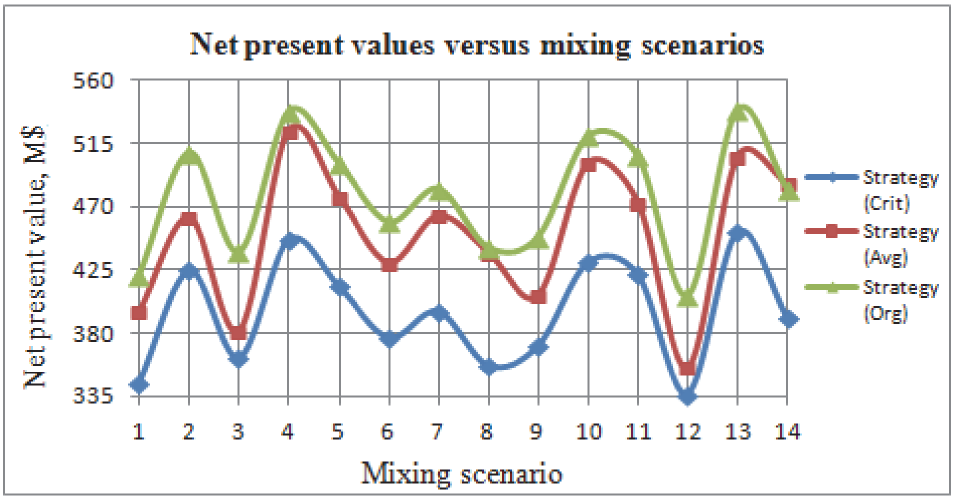


Fig. 70: Project net present values versus mixing scenarios for the three strategies.

It could be observed from the figure that all of the three methods paths are almost taking the same trend, with different magnitudes. This trend is in fact due to the different scenarios contribution and its resulting respective annual physical properties. In other words, this trend is reflecting the energy consumption expenses conditioned with the final concentrated tonnage due to the final milled ore.

It could be seen from the figure that the (Crit) method shows the least values, although it was previously showing the highest values of the expected metal recovery.

This may arise the importance of the stronger effect of the expenses regarding this project, which suggests the (Crit) method, compared with the others. This mainly due to the more energy consumption required to give the previous highest metal-grains liberation.

It could be noticed also from the figure that the (Org) method shows the best of the all methods, across the whole scenarios. That is mainly due to the combination between its comparatively high expected metal-recovery and the consumption of the exact needed grinding energy, for each individual ore-type across the years of the project life.

Environmental results and overall assessment for the arrangement methodologies

After calculating of the different stages power consumption for the three plant arrangement methods inside the modified model functions, the corresponding resultant emission of the green house gases is also estimated. These emissions are, in fact, calculated according to the conversion factors, which convert these electric power units into their corresponding amount of exhaust emissions at the power station during generating this energy.

The specific CO_2 emission (due to plant) for the whole suggested scenarios for the (Avg), the (Crit), and the (Org) arrangement methods are found in Table Ap4-8, Table Ap7-8, and Table Ap10-6, respectively. The summation of the CO_2 emission due to the electric energy consumption inside the plant (crushers and mills) and the fuel consumption, due to the loading and hauling operations, give the overall annual produced tonnage of the resultant gas emissions for the whole project.

Considering of the following concepts:

- More CO_2 emissions means more expenses required, in order mitigate these greenhouse effects on the atmospheric environment;
- More metal recovery maintains the sustainability concept due to preserving of the mineral resources; and decreasing of the valuable metal wastes and the contaminated tailings; and
- More realized net present value means higher economic benefits and real profitability,

a comparable multiplication factor is included to each scenario calculations. This multiplication factor is designed to express a global assessment to each scenario state. By this way, the differentiation between special scenarios, which have rather convergent economical values, for example, but divergent environmental effects, can be facilitated.

This assessment factor (K_{ass}) is, simply, the quotient of the expected metal recovery amount, $M_{tot.rec}$, multiplied to the net present value and divided by the resultant amount of the total project CO_2 emissions, $M_{tot.CO2}$; and is called shortly the (*Eco-Eco Factor*) referring to the (Ecology-Economic Multiplier Factor) as followed:

$$K_{ass} = \frac{M_{tot.rec} \times NPV}{M_{tot.CO2}} \quad (101)$$

Figure 71 shows the results of the calculated (*Eco-Eco Factor*) for the whole suggested scenarios for the (Avg), the (Crit), and the (Org) arrangement methods. The difference between the three plant arrangement methods is now more obvious, despite their rather analogical trends.

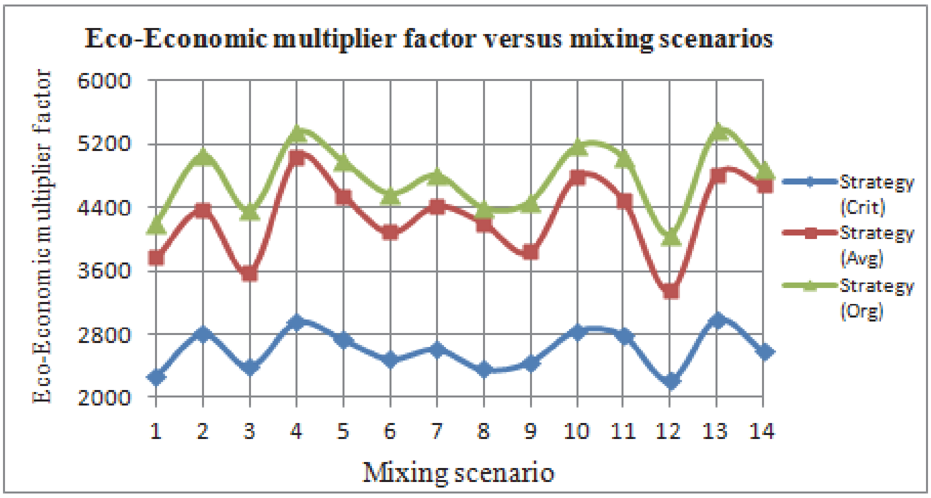


Fig. 71: *Eco-Economic* multiplier factor versus mixing scenarios for the three strategies.

Thus, on the bases of an integrated economic, sustainability, and environmental aspects, it could be concluded that the (Avg) method is better than the (Crit) method by more than 60 %. In the same time, the (Org) method, which belongs to the processing strategy (B), (section 6.3.2), is better than the (Avg) method by about 12 %.

The plant arrangement method preference, in addition to the interpretation for the different magnitudes, which appears between the mixing scenarios, within the same arrangement method, will be focused in the next section.

The Organized Method choice as the best strategy

Table 35 shows the concluded results which are mainly used, in order to differentiate and make the required preference to choose the best plant arrangement strategy and the best extraction and mixing scenario. The table illustrates the total power consumed for the crushing and grinding for each scenario of the three arrangement methods with their expected environmental impacts.

Table 35: Financial and environmental judgment data and preference for the three processing arrangements.
(Financials, M$; Power, GW; CO_2 emission, kt; Metal recovery, MIP; and (IRR and PI) are DMNL)

Sc./Item	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
Average Values Arrangement, (Avg)														
Consumed power	2683.985	2683.933	2684.130	2683.748	2683.827	2683.963	2683.673	2683.821	2687.015	2683.801	2689.253	2684.385	2683.835	2683.777
Metal recovery	3705.444	3672.853	3650.418	3735.159	3707.214	3691.674	3708.057	3736.008	3674.961	3719.669	3700.559	3664.621	3703.324	3747.684
Total CO_2 emissions	388.902	388.897	388.917	388.876	388.885	388.899	388.868	388.885	389.223	388.882	389.460	388.944	388.886	388.880
(NPV)	394.997	461.905	381.419	523.959	477.149	429.961	463.325	437.399	406.520	500.612	472.887	355.427	505.062	485.787
(IRR)	0.449	0.557	0.440	0.626	0.565	0.504	0.545	0.507	0.472	0.595	0.562	0.394	0.619	0.566
(PI)	2.944	3.273	2.877	3.579	3.348	3.116	3.280	3.153	3.001	3.464	3.327	2.749	3.486	3.391
Critical Ore-type Arrangement, (Crit)														
Consumed power	4435.350	4435.350	4435.361	4435.341	4435.340	4435.355	4435.341	4435.343	4435.371	4435.340	4435.395	**4435.366**	4435.342	4435.340
Metal recovery	3788.104	3788.077	3788.205	3788.077	3788.080	3788.204	3787.720	3788.086	3788.980	3788.078	3788.525	**3788.413**	3788.078	3788.030
Total CO_2 emissions	574.546	574.546	574.547	574.545	574.545	574.546	574.545	574.545	574.549	574.545	574.551	**574.548**	574.545	574.544
(NPV)	344.557	424.727	362.582	447.011	413.951	377.029	396.036	357.034	370.946	430.893	422.148	**336.604**	452.474	391.428
(IRR)	0.423	0.541	0.445	0.569	0.519	0.469	0.494	0.446	0.457	0.544	0.530	**0.404**	0.591	0.488
(PI)	2.696	3.090	2.784	3.200	3.037	2.855	2.949	2.757	2.826	3.121	3.077	**2.657**	3.227	2.926
Organizing Arrangement, (Org)														
Consumed power	2957.427	2957.366	2957.660	2957.368	2957.364	2957.662	2957.285	2957.391	2960.347	2957.369	2963.972	2957.930	**2957.369**	2980.120
Metal recovery	3764.127	3764.098	3764.236	3764.103	3764.099	3764.235	3763.749	3764.113	3768.961	3764.098	3770.526	3764.439	**3764.101**	3764.056
Total CO_2 emissions	388.693	388.687	388.716	388.687	388.688	388.715	388.703	388.689	388.682	388.688	388.744	388.726	**388.687**	372.908
(NPV)	421.517	508.284	437.926	537.889	501.021	459.293	482.833	441.263	448.560	519.923	506.158	407.157	**539.003**	482.633
(IRR)	0.479	0.610	0.499	0.647	0.590	0.533	0.564	0.512	0.516	0.619	0.600	0.450	**0.665**	0.563
(PI)	3.074	3.501	3.155	3.647	3.466	3.260	3.376	3.172	3.207	3.559	3.491	3.004	**3.653**	3.375

It also concludes the total expected metal recovery and the final cumulative net present value, with other economic factors such as the internal rate of return (IRR) and the profitability index (PI) for each suggested project strategy.

This table is used with Figure 72, in order to illustrate the strategy preference and scenario choice, based on the previously mentioned (*Eco-Eco Factor*). As the (Avg) method uses the mean values for all of the physical and chemical parameters, it can be considered just as a guide and not a reliable method to be used in installing the project. The (Crit) method is more pragmatic than the (Avg) method; therefore it will be used in comparison with the other pragmatic processing arrangement method (Org).

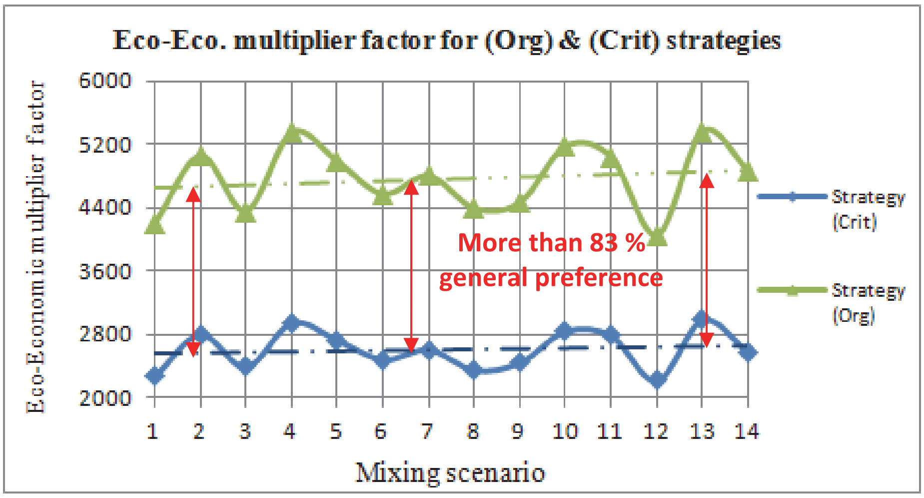


Fig. 72: *Eco-Economic* multiplier factor difference between the (Org) and (Crit) strategies.

Because of that, theses two arrangement methods are belonging to the previously designed processing strategies (A and B), this comparison can be considered as a real differentiation between these two strategies, themselves.

The figure shows a general preference for the (Org) method, across all of the different mixing scenarios, compared with the (Crit) method, by more than 82 % factor. This is due to the substantial difference between the two plant facilities arrangement concepts for them.

Hence, it can be concluded that the dealing with each ore-type, separately, with its corresponding set-points and own production line, in paralleled with the other corresponding production lines, will result in the best optimization for the plant operations. This will compensate for the rigidity (un-flexibility) for the plant response for optimization, especially at the fine grinding stage, which was encountered before, when experimented to be optimized in the first phase of the model instillation and the project investigation.

The final figure of the (Org) scenarios path declares the preference of the three scenarios 13, 4, and 10, which have the ore-types shares of (20 % 60 % 20 %), (30 % 50 % 20 %), and (30 % 45 % 25 %), respectively. Referring to the main physical and chemical characteristics for each individual ore-type (CH.5), it is indicated that the ore-type (1) is considered the softest one, while the ore-type (2) is the richest and the ore-type (3) is the intermediate in the metal content and also in its existence share within the mineralization area.

Hence, for the first approximation, it can be concluded that the metal content, the hardness magnitude, and the natural existence share of the individual ore-types, can be the reason for this differentiation.

6.6.5 Comparison between scenarios

In order to clarify the real output diversion, which led to the preference of the mining and processing strategy (A) on the other strategy (B), a comparison is made between the two scenarios (Crit-12) and (Org-13).

Figures 73-80 show an illustrated comparison between the two scenarios, through presenting of their most important outputs, which can be reviewed through their corresponding tables in the appendices.

From the figures, it can be noticed that (Crit-12) is always having about 50 % more energy expenses than that of (Org-13), which is confirmed by the same percent of the cumulative consumed power at the project end-life for both of them.

The higher income for (Crit-12), which is indicated after the 8^{th} year until the 15^{th} year, is just mitigating the difference between the slopes of the cumulative net present value for the two strategies, which is always higher for (Org-13), with an obvious stepper trend, than (Crit-12) at the first project years.

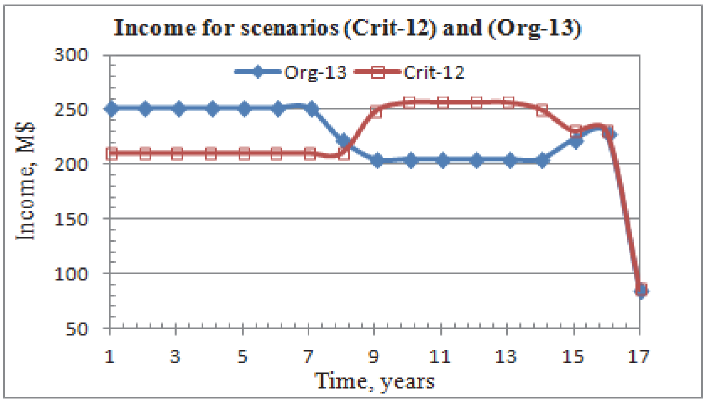


Fig. 73: Income comparison between scenarios (Crit-12) and (Org-13).

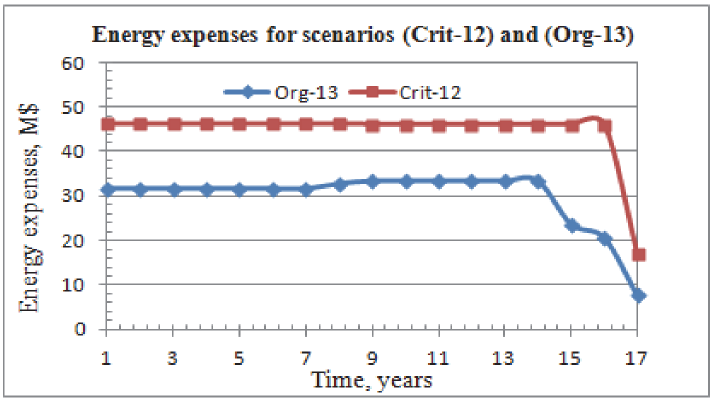


Fig. 74: Energy expenses comparison between scenarios (Crit-12) and (Org-13).

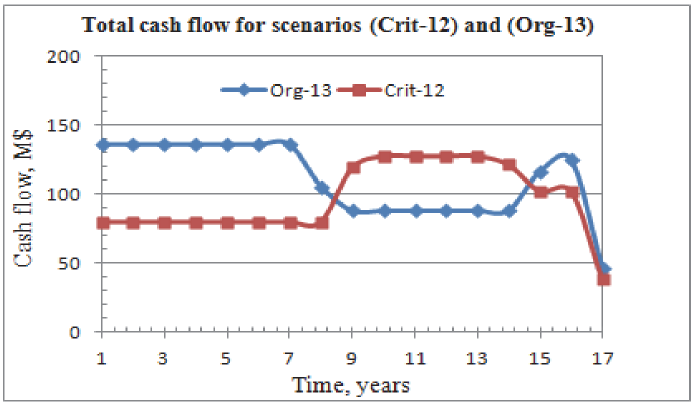


Fig. 75: Total cash flow comparison between scenarios (Crit-12) and (Org-13).

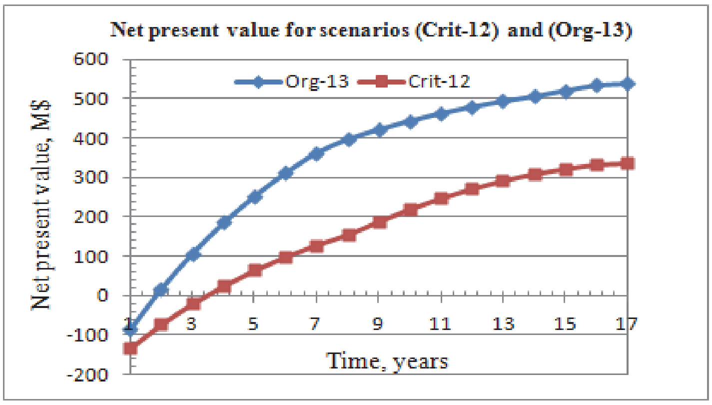


Fig. 76: Net present value comparison between scenarios (Crit-12) and (Org-13).

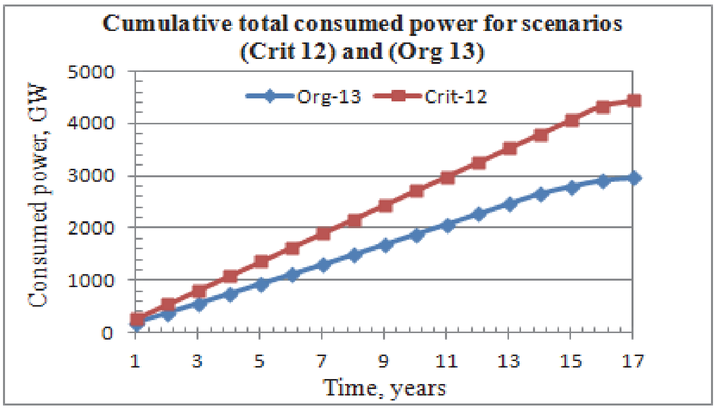


Fig. 77: Cumulative consumed power comparison between scenarios (Crit-12) and (Org-13).

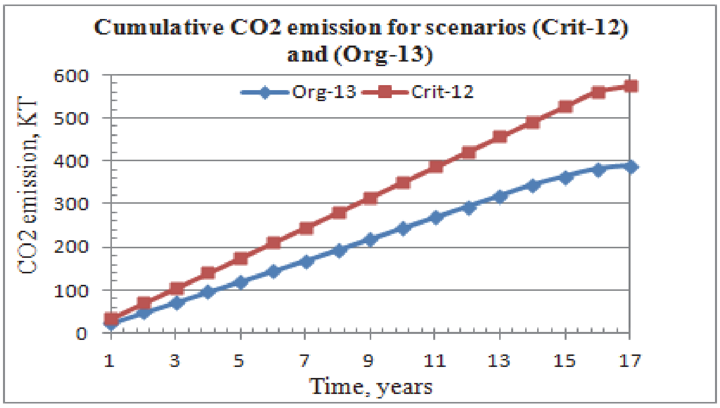


Fig. 78: Cumulative CO_2 emission comparison between scenarios (Crit-12) and (Org-13).

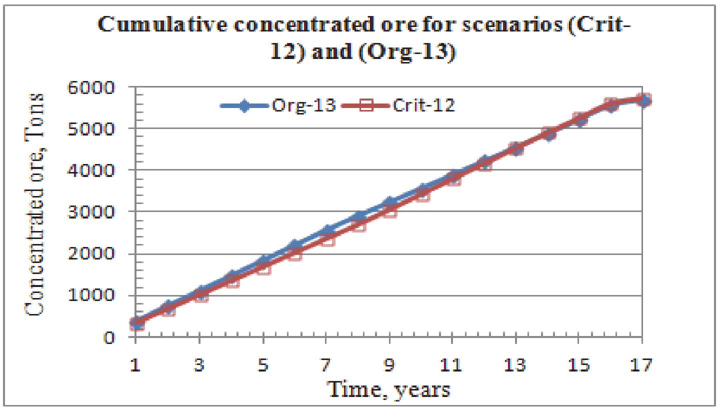


Fig. 79: Cumulative concentrated ore comparison between scenarios (Crit-12) and (Org-13).

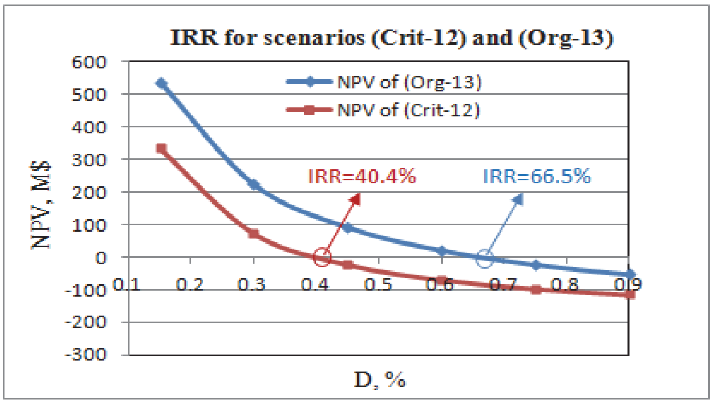


Fig. 80: Internal rate of return (IRR) for scenarios (Crit-12) and (Org-13).

The net present value for the project, which is suggested by the (Org-13) method, is higher than that corresponds to (Crit-12) by more than 60 % (539.003 M$ versus 336.604 M$), (Table 35), in addition to giving almost the same expected metal recovery, which less than (Crit-12) by just 0.6 %. Moreover, it is indicated from the figures that (Crit-12) is producing about 48 % more cumulative tonnage of the CO_2 gases than that of (Org-13).

All of the previously mentioned better results for (Org-13), compared to those of (Crit-12), give the preference to the former method by more than 140 %, according to their combination through the *Eco-Economic Multiplier Factor*.

Finally, and from a merely economic point of view, the internal rate of return for the suggested project methodology realizes about 65 % more preference for the (Org-13) method on the other corresponding to (Crit-12).

Figures 81 & 82 show an example for the mass flow characteristics, at the 7th year of the mine life, for both of the plant facility arrangement scenarios (Crit-12) and (Org-13), respectively.

The figures show the tonnage feeding rates, the tonnage by-passes, and the accompanied ore grain size, according to the different set-points, for the two methods (Crit-12) and (Org-13), across the consecutive stages of the ROM size reduction and processing.

It is noticed that, at this seventh year, the cumulative present value of the (Crit-12) scenario is 35.4 % of that of the (Org-13) scenario and the CO_2 cumulative emission is 68.7 % of that for (Crit-12). In the same time, the cumulative (30 % conc.) ore for (Org-13) is 8.7 % more than that for the other method. It should be mentioned that at the end life of the mine, these values become 62.5 %, 67.7 %, and (-0.6 %).

6.6.6 Extreme cases versus the chosen *Organized Method*

In order to examine what is the principal effective parameter, which causes this fluctuation between the different scenarios, for about 30 % difference between the upper and the lower scenario, within the chosen (Org) method itself, three new more scenarios, each of which represents an extreme case, are investigated.

The 1st is called scenario (A), which suggests beginning the mining and processing for the richest ore-type, for the first years until being damped, then extraction of the 2nd higher grade one and so on. Sure this will causes the acceleration and the increasing for the annual income.

Thus, this lets us examine if the higher expected net present values, due to the accelerating for the higher annual income at the first years of the project life, can be the real responsible for the best final results.

The 2nd extreme scenario is called scenario (B), which suggests beginning mining and processing for the ore-type of coarser texture grain size, for the first years until being damped, then extraction of the 2nd one and so on. This beginning with the softest ore-type will assure decreasing in the energy expenses and reduce the annual energy cost.

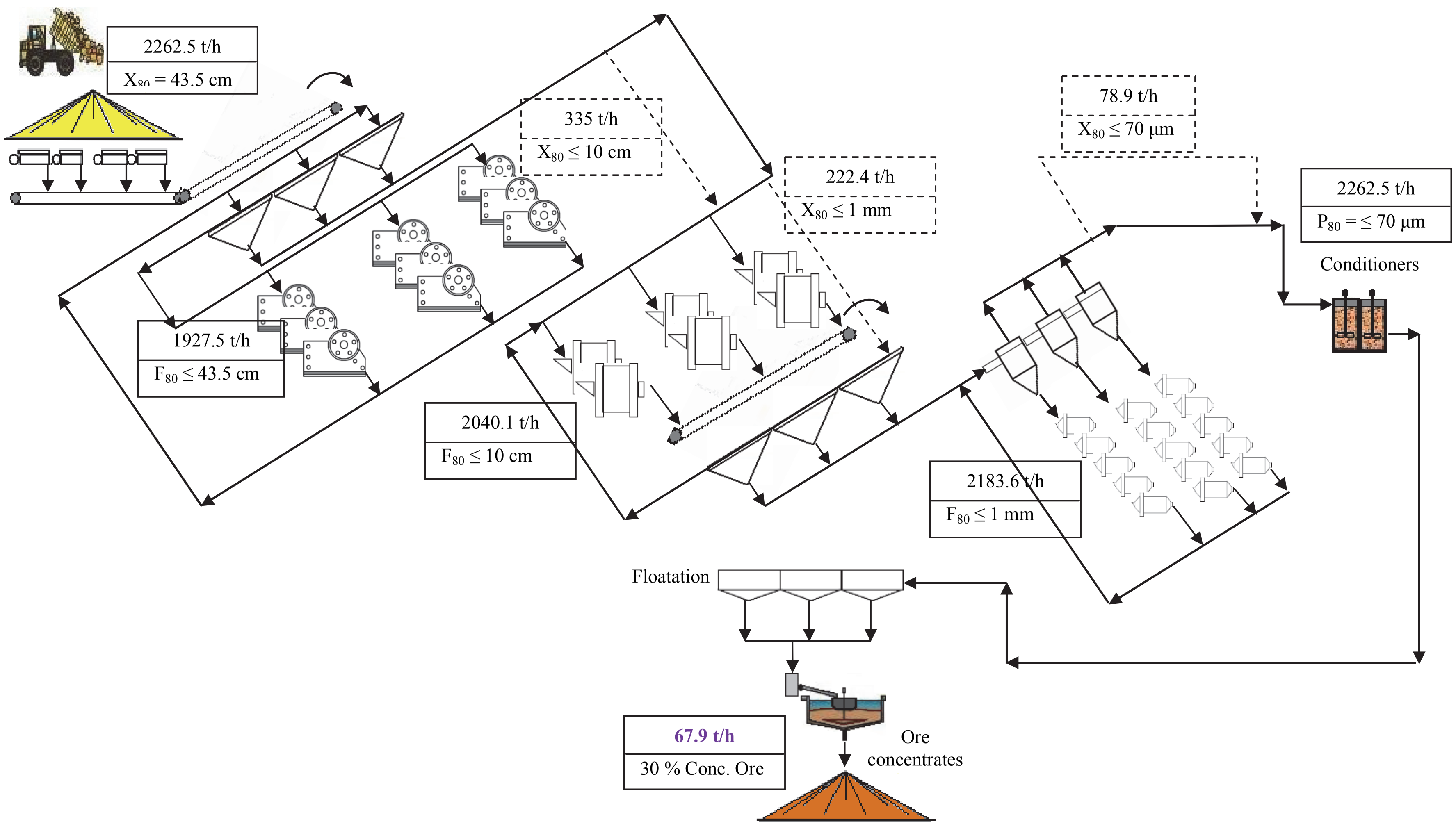


Fig. 81: An example for the mass flow characteristics for the scenario (Crit-12) at the 7th year of the mine life.

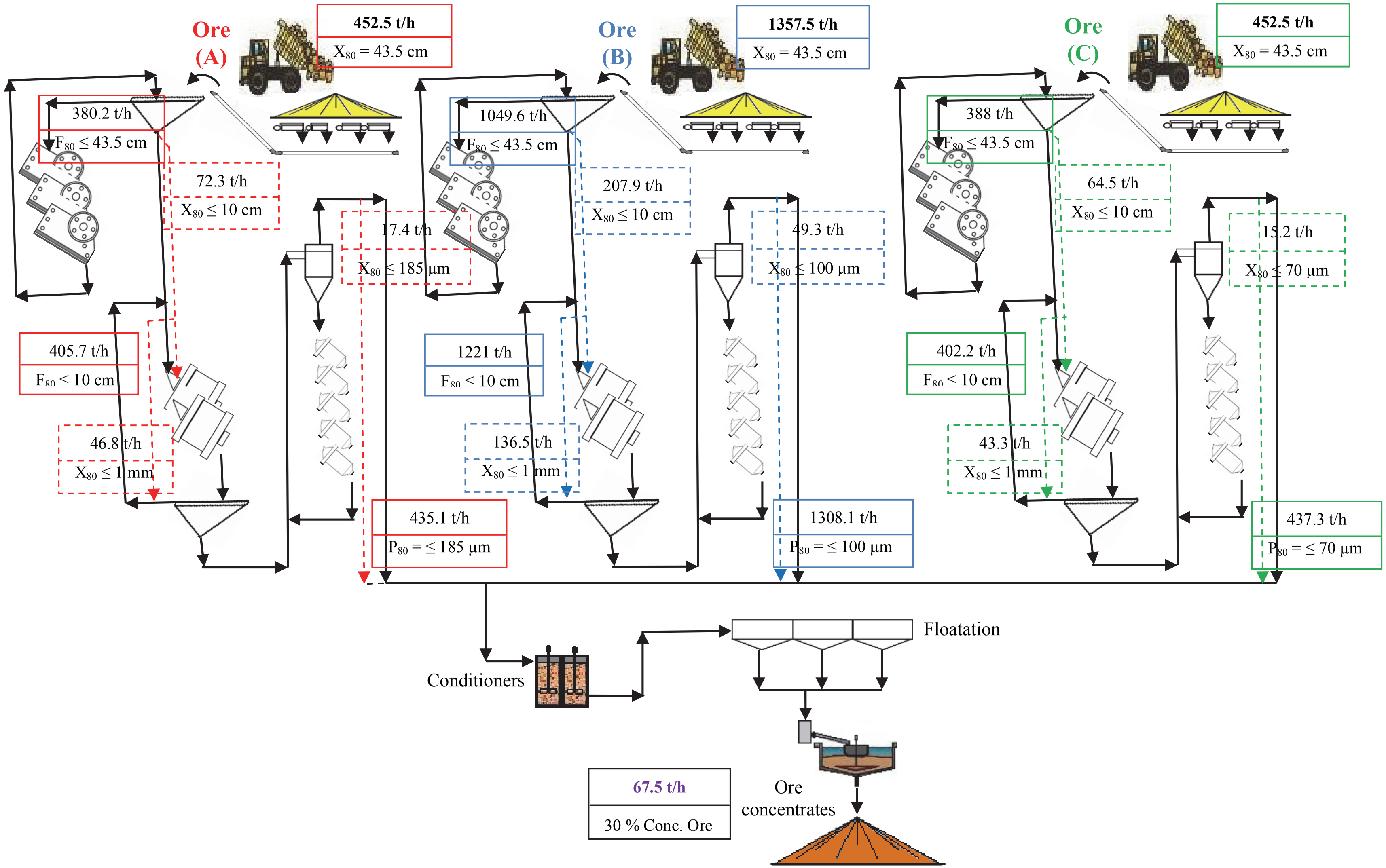


Fig. 82: An example for the mass flow characteristics for the scenario (Org-13) at the 7th year of the mine life.

The 3rd extreme scenario is called scenario (C), which suggests beginning mining and processing for the ore-type of the hardest one, for the first years until being damped, then extraction of the richest ore-type, then, afterwards, the remainder one. The remainder series alternatives, D, E, and F are also considered, in order to give the possibility to compare the parallel scenarios with the all available series ones.

Figures 83 illustrates the project net present values while Figures 84 illustrates the assessment factors for all of the (Org) mixing scenarios, which are drawn together with the corresponding values for the series and the Extreme cases scenarios.

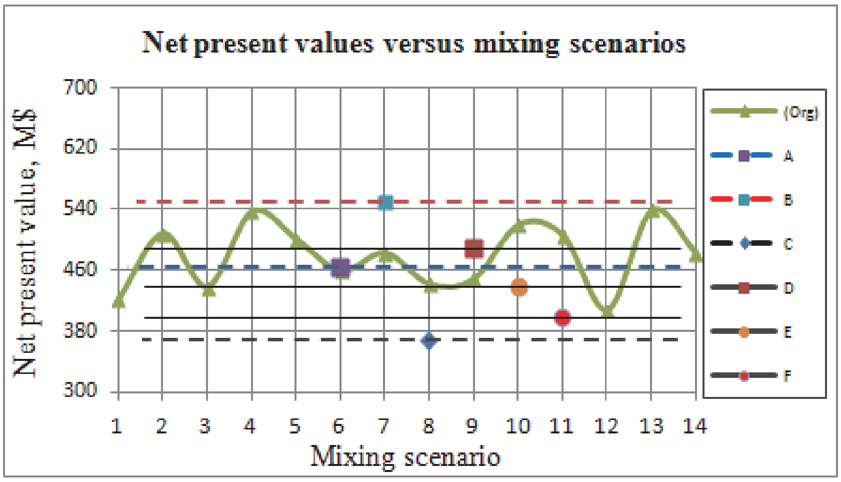


Fig. 83: Project net present values for the (Org), series and the Extreme cases scenarios.

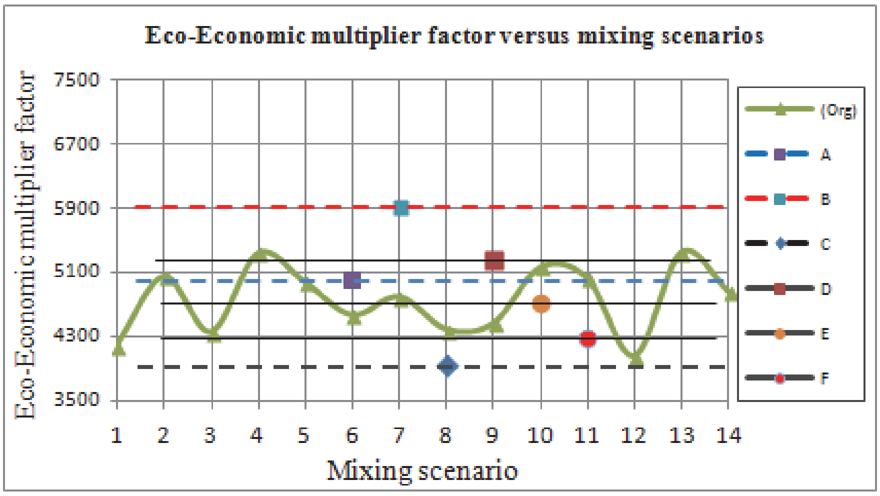


Fig. 84: Assessment factors for the (Org), series and the Extreme cases scenarios.

From the figures, some observations can be outlined as follows:

- The focusing on the higher grade ore-types to be extracted firstly, (Case A), is not the pioneer in this context, although it shows somewhat fair results, as the more effective here is not by increasing the income alone, but moreover the reducing of the energy expenses at the first years of the mine life.
- The 3rd extreme case, (Case C) shows, comparatively, the least values because it started by the hardest ore-type, despite it is followed by the richest one. This indicates that the extraction of the richest ore-type can not mitigate the highly bad effects of starting by the most energy consumable one.
- The 2nd extreme case (Case B) is representing the pioneer and the ideal scenario. That is because the beginning with the coarser ore-type, which represents the one of the least hardness, leads to decreasing the annual energy costs and the accompanied CO_2 emissions at the first period. The little reduction in the income at this period has not any bad effects, compared to the other stated positive effects. Thus the NPV and the *Eco-Economic* factor for this scenario declare the optimality of considering it.
- In the same time, three mixing scenarios 13, 4 and 10 are better than the extreme case A, while all the mixing scenarios are better than the extreme case C.
- The other series intermediate alternative cases, which are D, E, and F, are all lying between Extreme cases B and C.

Finally, it can be conclude that the integrated combination for the metal content (grade), the hardness magnitude (rigidity), and the natural existence share for each individual ore-type has a great effect on the optimal scenario choice, with a high priority to the ore-types texture grain size and hardness.

6.7 Optimization evolution overview across the operations improvement steps

The optimization evolution, due to the integrated mining and processing improvement by the operations modeling and simulation, is illustrated in Figure 85.

The figure shows an overview of the optimization steps, which are indeed the consecutive cumulative improvement methods for the mining and processing sequential operations, starting by drilling and blasting and ending by shipping to the smelter.

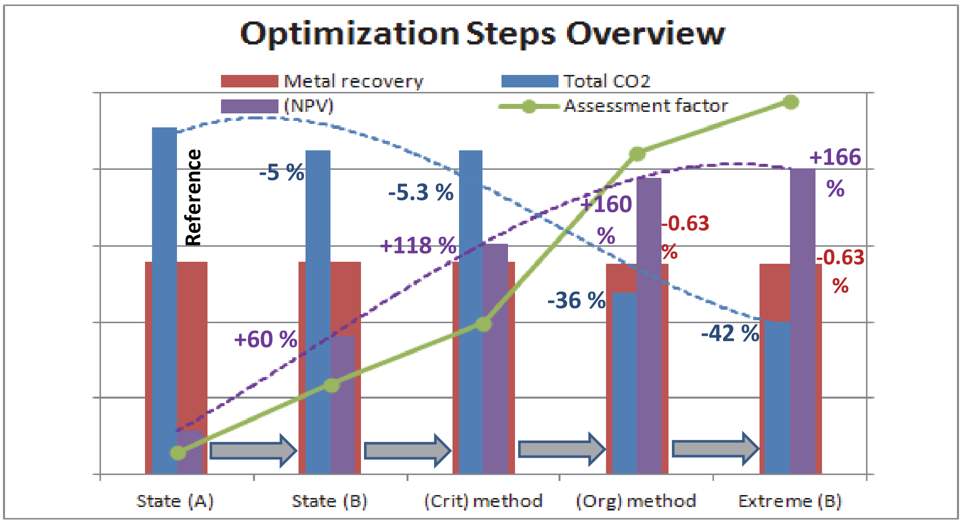


Fig. 85: Optimization steps overview and methods preferences.

As shown in the figure, and starting by the previously suggested reference state (State A), the first optimization was by transferring to mining strategy (State B) by choosing and applying of the optimal blast fragmentation size. This causes an obvious improvement in the loading and hauling operations and also with the primary crushing stage, while the effects on the grinding stages are untouchable. This improvements result in an increasing of the NPV by 60 % and decreasing in the total produced CO_2 tonnages by 5 %, without any loss in the metal-recovery.

By further optimization, through the more deeply investigation for the mineralization area and the application of the selectivity in mining by introducing special extraction scenarios for the existing different ore-types, more improvements are experienced by transfer to the (Crit) method. The improvements within this method are by reducing the drilling and blasting costs, due to the selectivity in the ore mining and extraction. This improvement had also its effects on the plant operations, with further improvements in the different size reduction stages. These improvements result in an increasing of the NPV by 118 % and decreasing in the total produced CO_2 tonnages by 5.3 %, without any loss in the metal-recovery.

The next step in the operations improvements is by transferring to the (Org) method. These improvements are concentrated in the processing stages inside the plant by using the upper determined cumulative best results. This method, which is charged to examine the different ore-type mixing scenarios through the designed assigned parallel separated production-lines, yields an obvious better optimization. This step, which represents the result of the global relating between the different mining and processing operations, resulted in an increasing of the NPV by 160 % and decreasing in the total produced CO_2 tonnages by 36 %, while the loss in the metal-recovery was just 0.63 %.

The last step in the operations improvements is inspired from the good results, which belong to the upper mentioned (Org) method. The transfer to the case (Extreme B) gives the overall best optimization for the integrated mining and processing operations.

This step resulted in an increasing of the NPV by 166 % and decreasing in the total produced CO_2 tonnages by 42 %, while the loss in the metal-recovery was also just 0.63 %.

In the same time, the final results of this last method declare an important conclusion belonging to the optimal used method in mining and processing of the multi-metals ore deposits, such as the copper-porphyry deposits.

Although the integrated combination for the metal content (grade), the hardness magnitude (rigidity), and the natural existence share for each individual ore-type, has a great effect on the optimal chosen method for mining and processing these deposits, the high priority should be for the ore-types texture grain size and hardness.

The different parts of mineralization area should be managed and planed according to their hardness and mineral liberation grain size, as these parameters have the highest effects on the energy consumption inside the plant, the total produced CO_2 emissions and, hence, the total costs and the final feasibility of the project.

Figure 86 illustrates an overview for the ore deposit mining and processing scenarios by the parallel methods, which are the fourteen scenarios for the (Org) plant arrangements, and by the series methods, which are the (Extreme) scenarios for the plant organization. The previously mentioned best two scenarios are outlined by the red rectangles, from which the first is the (Extreme B) and the second is the (Org-13).

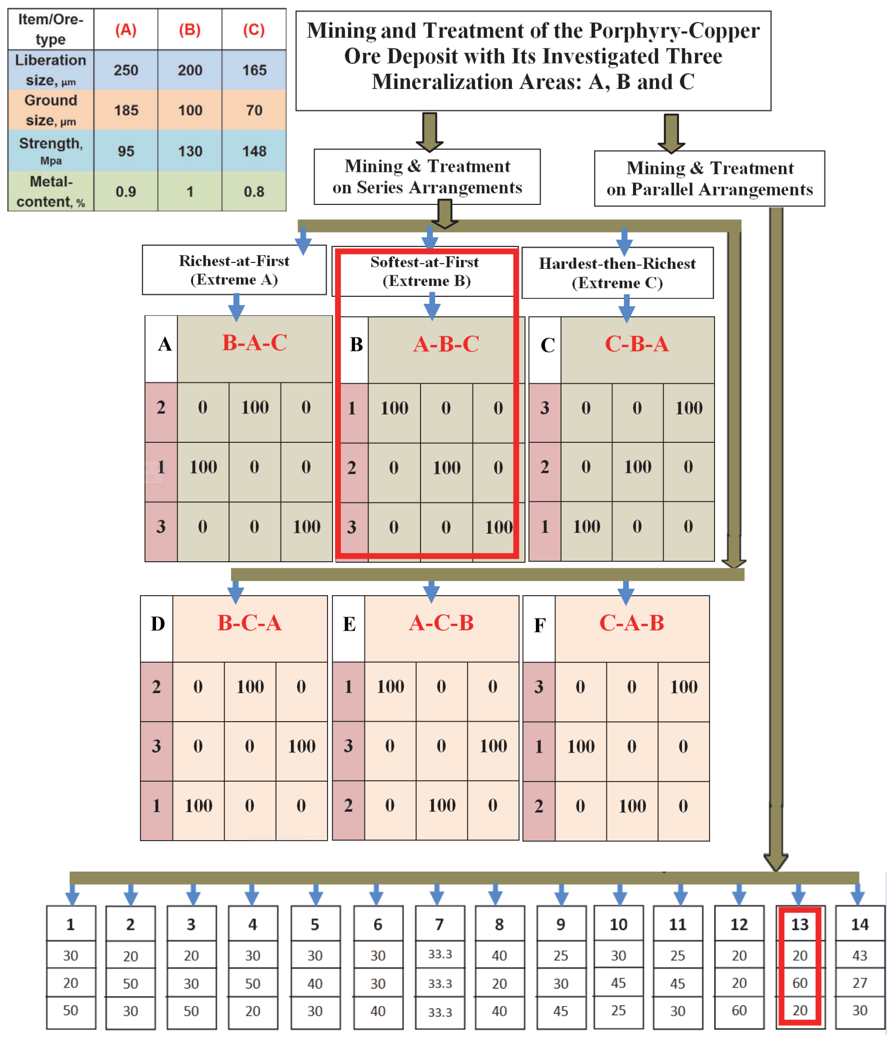


Fig. 86: Outline for the ore deposit mining and treatment scenarios by parallel (Org) and series plant organization methods.

It is mentioned also from Figure 87 that more than 63 % of the achieved optimization is obtained due to the special organizations for the plant size reduction facilities (parallel and series), while the other 37 % is obtained from the all other improving measures, such as the fleet and the optimal fragmentation size choosing, as well as the mining selectivity application. This illustrates the importance of considering the special planning and arrangement for the plant facilities and production-lines, in the same time when planning and scheduling the extraction blocks from the different mine areas, according to the ore-types physical prosperities specially hardness, texture grain size, and metal contents.

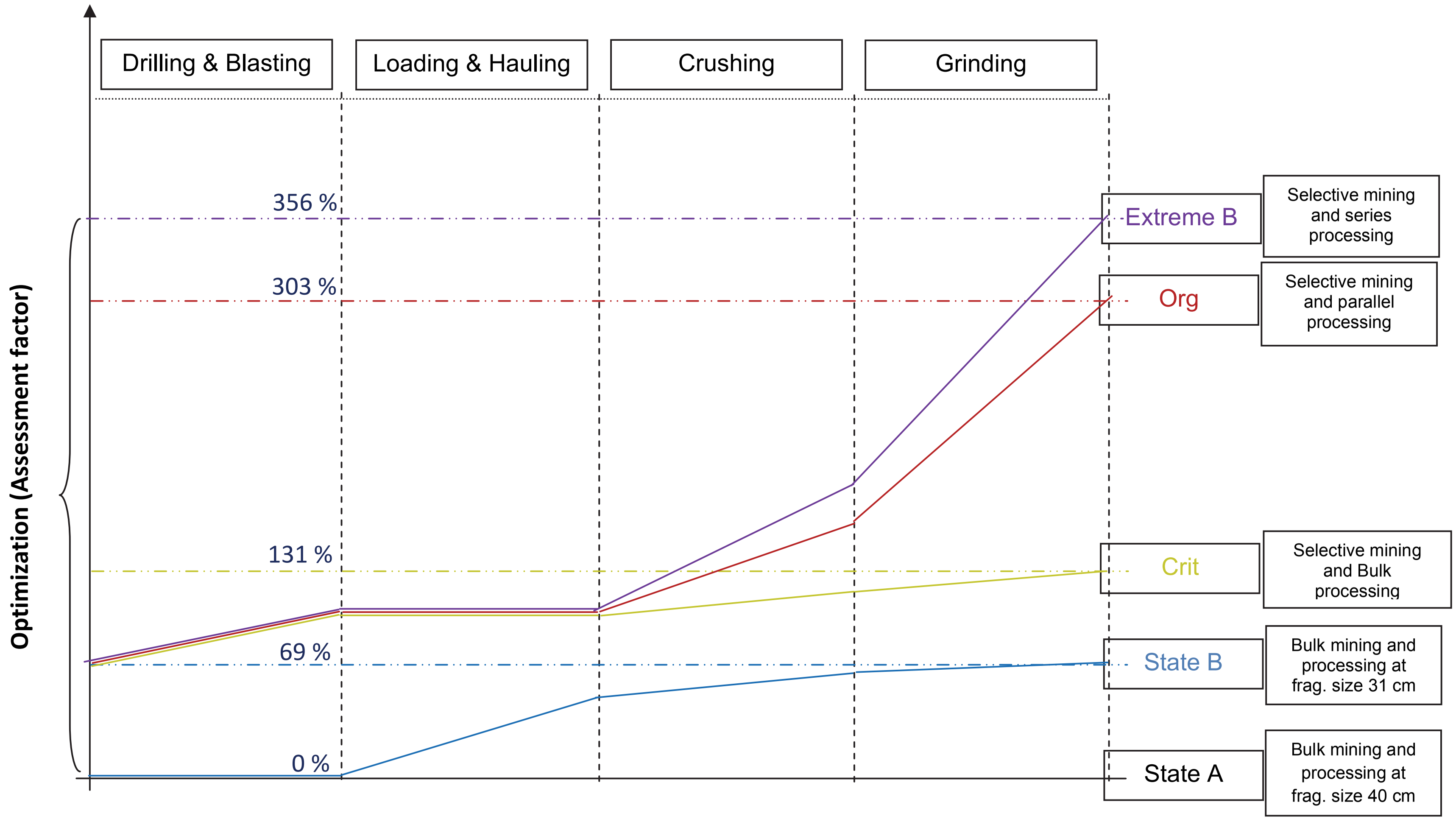


Fig. 87: Mining and processing optimization evolution according to the different operations and the adopted methods.

7. Conclusion and Recommendations

Conclusion:

1. Low-grade mineral deposits lead to a very high tonnage excavation with the adherent economical and environmental problems, such as the higher machinery gas emissions; and lead also to high costs of the valuable products recovery. Moreover, due to the very high operational, maintenance, spare parts, equipments and processing costs, mineral commodities, especially metals, show an increased trend in their prices. This is accompanied with the higher demand for the mineral resources, especially metals, due to the higher industrial requirements and the higher social levels.
2. These challenges can be overcome through mine planning optimization, in order to realize the economic benefits of the required ore production with the best quality and the lowest costs. Therefore, an approach for the global optimization of the integrated mining and processing operations is designed through the mining selectivity strategy, the deeply investigation of the ore deposit parameters, and the proper adaptation and planning for the plant facilities. This is done mainly through construction of a dynamic modelling and simulation for the whole mining and processing sub-operations, by using the (VENSIM-PLE) software.
3. The suggested approach presents most of the mining and processing stages. It addresses the flexible operations, which are affected greatly with the previous operations or have certain influences on the subsequent ones, and achieves the suitable modifications within them. Afterwards, further modifications, through special organizing scenarios, are done in order to achieve the overall mining and processing optimization.
4. The model is constructed with its (*Reference*) mode, which is after further optimizations of it to a (*Control*) one, was containing 310 parameters, 13 lookups, and more than 100 algebraic equations and formulas. The model, which covers the operations from drilling to shipping the concentrates to the smelter, is divided into three sub-models: *Drilling and Blasting*; *Loading and Hauling*; and *Crushing and Grinding*, with inter-connecting and linking between them to form a coherent system-model for the global operations. A copper-porphyry deposit, with three different

ore-types, is selected to be a case study data for using in the modelling construction and discussion.

5. The mining and processing organization of the ore body was the core of the model optimization, after constructing of its reference mode, by providing an online selectivity in mining and design of different scenarios for the processing production lines organization. Introducing of different scenarios for the extraction selectivity, according to the different portions of each ore-type within the mineralization area, examined 14 scenarios for parallel organization methods for the plant production lines and 6 more scenarios for the series organization. Each scenario had a certain concept for how to process the three different ore types with a continuous, parallel, and separate technique, which depends on special organizations using pre-grinding blending (Strategy A) or post-grinding blending (Strategy B), in order to realize the maximum mineral recovery and the minimum fine grinding costs.
6. A special coding triangle for the three ore-types blending is designed, in addition to an *Excel-File* calculation tool, in order to calculate the annual tonnage, production shares, and ROM feed prosperities belonging to each scenario. The outputs of the calculation tool are introduced to the optimized (*Control*) mode of the model as inputs through special lookups. The different scenarios are applied to three data-set and calculation methods: one uses the average data for the characteristic parameters and the production lines set-points (Avg), the other uses the data belonging to the most critical ore-type (Crit), and the third uses the characteristic parameters and the production line set-points for each ore-type, individually.
7. Specified simulations for the model with a total number of 11 different runs, are made, in order to examine the overall operations costs by changing the blast fragmentation size from 15 to 65 cm, with a first assumed reference state (State A) equals 40 cm. The results showed that the best economic range of using the specific explosive energy is between 0.8-1.2 kg/m^3. Thus, by conducting more runs within the chosen range of the specific explosive energy with narrow intervals, the preliminarily results show good and reliable optimization within the extraction and the transportation operations, by choosing the optimal fragmentations size of 31 cm (State B), with a corresponding powder factor equals 1 kg/m^3 and calculated mine life equals 16.4 years, and choosing of the most suitable loading and transporting fleet, which was in this case (Fleet A).

8. As a fundamental test for the model robustness, a sensitivity analysis to it is done. This is made considering the final choice for the optimal mean blasting fragmentation size. Different cases, which assumed changing in the fuel and electric energy prices from -10 % to + 50 % of their values, are made. The effect on the original magnitude for the optimal fragmentation size was just by (+ 1.6 % to -6.1%), which lies in the range of the accepted statistical error.
9. Choosing of the optimal fragmentation size and the suitable fleet optimized the extraction and the transportation operations through an 8.53 % reduction in its own specific cost. Regarding to the plant optimization, the primary crushing stage could be also be optimized by 12.5 % energy cost reduction, while the other stages showed no high optimization, with 2.69 % and 1.13 % reduction of the costs of the coarse and fine grinding, respectively.
10. The main source for the cost optimization through the preliminarily results was nearly the (*Loading and Hauling*) operation, which forms more than 31 % of the total project cost. Although the fine grinding process was forming more than 75 % of the total energy consumed within the processing plant and about 25 % of the total project costs, its optimization, within the reference mode, was obviously low.
11. Within jaw crushers, the limiting factor is the loading degree, which reaches to more than 85 % of the total available facilities, while the power limits play much less action with a maximum magnitude reaches not more than 33 % of its total availabilities. Within coarse grinding, the corresponding values are 64 % and 76 %, while with the fine grinding stage, the corresponding values reach their maximum, which are 89 % and 86 %. This spots light on the critical situation for the fine grinding stage. An important reason for this is that the natural characteristics and parameters belonging to the ore-type, hardness, micro-textures, liberation size,…etc, are prevailing at this stage, not the other effective operational and technological factors, such as the other operations.
12. Regarding the results after the model optimization, the metal-recovery rate for the (Crit) method gives the highest metal recovery, which is just about 0.6 % more than that of the (Org) method, while reaches to about 3.8 % more than the (Avg) method. It is noticed also that both of the ((Crit) and the (Org) methods gives fixed metal recovery production, while with the (Avg) method was taking the figure, which is resulting from the reaction between the different annual ROM delivery to the plant and the used average values of the plant set-points. On the bases of an

integrated economic, sustainability, and environmental aspects, it could be concluded that as the (Avg) method is better than the (Crit) method by more than 60 %, the (Org) method was better than the (crit) method by more than 83 %.

13. The final figure of the (Org) scenarios path declares the preference of the three scenarios, which have the ore-types shares of (20 % 60 % 20 %), (30 % 50 % 20 %), and (30 % 45 % 25 %), respectively. The net present value for the project, which is suggested by the (Org-13) method is 539.003 M$ versus 336.604 M$ for the other (Crit), which is moreover producing about 48 % more cumulative tonnage of the CO_2 gases than that of (Org-13).

14. By consequent steps of optimization through the (*Control*) mode, especially within coarse and fine grinding stages, and comparing to the beginning case during constructing of the first (*Reference*) mode, the improvements are made across six steps and resulted finally in an increasing of the NPV by 166 % and decreasing in the total produced CO_2 tonnages by 42 %, while the loss in the metal-recovery was just 0.63 %.

15. It is concluded that, although the integrated combination of the ore-grade, the grain hardness magnitude, and the natural existence share for each individual ore-type, has a great effect on the optimal chosen method for mining and processing this deposit, the high priority was for the ore-types texture grain size and hardness; and the different parts of mineralization area should be planed according to their hardness and mineral liberation grain size, as these parameters have the highest effects on the energy consumption inside the plant, the total produced CO_2 emissions and, hence, the total costs and the final feasibility of the project.

16. It is indicated that, more than 63 % of the achieved optimization is obtained due to the special organizations for the plant size reduction facilities (parallel and series), while the other 37 % is obtained from the all other improving measures, such as the fleet and the optimal fragmentation size choosing, as well as the mining selectivity application. This illustrates the importance of considering the special planning and arrangement for the plant facilities and production-lines, in the same time when planning and scheduling the extraction blocks from the different mine areas, according to the ore-types physical prosperities.

Recommendations:

1. It is recommended not to deal with the feed to the mill as a fixed average input grain-size, without special investigations for the internal-fracture, microscopic texture and the liberation grain-sizes, as many subsequent technical errors can arise, that if this physically inhomogeneous feed is ground to a fixed final product grain-size, then parts of the ore will be over-ground, consuming more waste energy, and other parts will be not completely exposed, reducing the final income considerably by transfer an amount of the valuable mineral to the tailings.
2. Although this model realized the followings: economical extraction and transporting of the ore, providing the ROM the susceptibility to be processed in an economical manner, achieve the consistency in production, reduce the various operations costs, increase the milling recovery and profitability, consider of the environment aspects, decrease the downtimes, and reduce of energy consumption, horizons for further researches are still insisting.
3. As the time available for conducting this study is limited, more investigations should be conducted out of its scope. Proposed investigations should be conducted upon the model, especially which are considering the existence of other ore deposits with other different parameters and characteristics; also other researches should be conducted for ore deposits, which have different metals, and not just different types of the same metal.
4. For me and researchers, who intended to work on this model or continue optimizing of the integrated mining and processing operations, especially who intending the work with the same software, it is recommended to make an intensive sensitivity analysis for the model, which considers the model response for changing certain parameters (individually or together). From these parameters, it could be the blasting parameters, which belong to the blasting pattern; the financial parameters for the different expenses and the selling prices; the time parameters, which belong to the operating times for the mining activities and the activities inside the plant, etc.

References

1 Alarie, S. and Gamache, M., 2002. *Overview of solution strategies used in truck dispatching systems for open pit mines.* Int. J. of surface min., reclam. and env., vol. 16, P. 59-76.

2 Aler, J., et al., 1996. *Measurement of the fragmentation efficiency of rock mass blasting and its mining applications.* Int. J. Rock Mech. Min. Sci. & Geomech., Vol. 33, pp. 125-139.

3 Alruiz, M., et al., 2009. *A novel approach to the geo-metallurgical modelling of the Collahuasi grinding circuit*. Minerals Eng., 22, 1060–1067

4 Altindag, R., 2002. T*he evaluation of rock brittleness concept on rotary blasthole drills*. J. S. Afr. Inst. Min. Metall., vol. 102, P. 61–66.

5 Altindag, R., 2003. *Correlation of specific energy with rock brittleness concepts on rock cutting*. J. S. Afr. Inst. Min. Metall., vol. 103, P. 163–71.

6 Ardalan, K. and Kobos, C., 2003. *Cash flow implied interest rate: A unified approach*. Marist College.

7 Atlas Copco Blasthole Drills, 2009. *Total Drilling Cost AC BHMT's Value Proposition*. Prepared for Indiana Society of Min. and Reclam., 19th Annual Technology Transfer Seminar.

8 Atlas Copco Blasthole Drills, 2010. *Pit Viper 351 manual*. www.atlascopco.com/blastholedrills.

9 Badger, W. and Banchero J., 1985. *Introduction to chemical engineering*. McGraw-Hill, Inc, Singapore, P. 677.

10 Bayer, A.K. and Winkel, R.M., 2004. *Come to where the copper is-Modern ore mining in Chile*. Germany World of Min. - Surface & Under. 56, no. 5.

11 Beattie, N.C., 2009. *Monitoring-while-drilling for open-pit mining in a hard rock environment*. M.Sc. Thesis, Queen's University, Ontario, Canada.

12 Benoit, J. and Bothner, W., 2002. *Characterization of fractured-rock aquifers using drilling parameters.* Proceedings of the fractured-rocks aquifers, Colorado, P.13-15.

13 Bieniawski, Z., 1989. *Engineering rock mass classifications*. John Wiley and Sons, New York.

14 Bond, C., 1952. *The Third Theory of Comminution*. Mining Engineering, p 484-494.

15 Bureau of Labor Statistics, 2008. *2 Massachusetts Avenue*, United States.

16 Bye, A., 2006. *The strategic and tactical value of a 3D geotechnical model for mining optimization.* The South Afr. Inst.e of Min. and Metal., P. 97-104.

17 Casteel K., 2005. *More productive drilling and blasting: optimising feed supply to the crusher can cut total production costs, so can better machine utilisation rates*. African Review of Business and Technology.

18 Caterpillar Performance Handbook, Edition 29. Publication by Caterpillar Inc., 2001, Peoria, Illinois, U.S.A.

19 Caterpillar, 2011. *6040 FS Hydraulic Mining Shovel technical catalog*, http://www. cat.com/cda/layout.

20 Caterpillar, 2011. *6060 FS Hydraulic Mining Shovel technical catalog*, http://www. cat.com/cda/layout.

21 Caterpillar, 2011. *789C Mining Truck technical catalog*, http://www.cat.com/cda/layout.

22 Caterpillar, 2011. *793F Mining Truck technical catalog*, http://www.cat.com/cda/layout.

23 Chadwick, J. and Higgins, S., 2006. *US technology*. Int. Min. September, 44–54

24 Chanda, K. and Ricciarodone, J., 2002. *Long Term Production Scheduling Optimisation for a Surface Mining Operation: An Application of MineMaxTM Scheduling Software.* Inter. J. of Surface Mining, Reclamation and Environment, Vol. 16, No. 2, pp. 144-158.

25 Chang D.J., 2005. *Vehicle speed profiles to minimize work and fuel consumption*. J. transportation engineering, P. 173-182

26 Cramer, A., 2001. *The extractive metallurgy of South Africa's platinum ores*. Journal of Metallurgy, 14–18.

27 Cross, C., 2005. *A Rio Tinto view of collaborative research in Europe and the World*. 16th. Society of Mining Professors (SOMP), Ankara, Turkey.

28 Crowson P., 2003. *Astride Mining: issues and policies for the mining industry*. Mining Journal Books, London.

29 Cunningham, C., 1983. *The Kuz-Ram model for prediction of fragmentation from blasting*. Proc. 1st Int. Symp. on Rock Fragmentation by Blasting, Lulea, Sweden, P. 439-453.

30 Cunningham, C., 1987. *Fragmentation estimations and the Kuz–Ram model, four years on.* Proc. of the 2nd Inter. symp. on rock fragmentation, Keystone, P. 75–87.

31 Dagdelen, K., 2001. *Open pit optimization - strategies for improving economics of mining projects through mine planning.* 17th International Mining Congress and Exhibition.

32 Deere, D.U. and Deere, D.W., 1988. The *rock quality designation (RQD) index in practice*. In Rock classification systems for engineering purposes, ASTM Special Publication 984, 91-101.

33 Denkhaus, H.G., 2003. *Second reply to R. Altindag, "Brittleness and drillability".* J. S. Afr. Inst. Min. Metall., vol. 103, no. 8, P. 527

34 Dhillon, B.S., 2008. *Mining equipment reliability, maintainability, and safety*. Hand book, Springer-Verlag

35 Dick, R.A., and Olson, J.J., 1972. *Choosing the proper borehole size for bench blasting*. Min. Eng., V 24, No.3, P. 41-45.

36 Dick, R. A., 1975. *Recent blasting fatalities in metal-nonmetal mining.* Pit and Quarry, V. 67, P. 85-87.

37 Ding, B., et al., 2007. *Defining the production scale of an underground mine.* Int. J. Miner. Resour. Eng., 12 (1), 1-19.

38 DOERING GmbH. Sinn, Germany, www.doering-sinn.de

39 Drebenstedt, C. and Kressner, M., 2009. *Integriertes Rockmanagementsystem (RMS) zur Kostenreduktion und Umweltentlastung im Bergbau.* Project report with Siemens co., (feasibility study), not puplished. TU Bergakademie Freiberg, Deutschland.

40 Drebenstedt, C. and Kressner, M., 2010. *Integriertes Rockmanagementsystem (RMS) zur Kostenreduktion und Umweltentlastung im Bergbau.* Project report with Siemens co., (Solution development-part 1), not puplished. TU Bergakademie Freiberg, Deutschland.

41 Dyno Nobel Asia Pacific Limited, 2011. *Blasting and Explosives Quick Reference Guide*. P. 14-19.

42 Edgar, J., 2011. *Brake Specific Fuel Consumption- A really useful concept.* AutoSpeed.

43 EERE, 2006. *Energy Efficiency and Renewable Energy. Using Mine-to-Mill Technology to Optimize Crushed Stone*. USA.

44 Elber, L., et al., 1994. *Development of a control system for a SAG-B-C grinding circuit.* Minerals Engineering, Vol. 7, pp. 153-167

45 Eloranta, J., 1995. *Selection of powder factor in large diameter blastholes*. EXPLO 95 Conference, AusIMM, Brisbane, PP 25-28.

46 Eloranta J., 2002. *The Role of Blast Operations in Metal Mining*. International Society of Explosives Engineers. Volume 1, P 45

47 Engin, I.C., 2009. *A practical method of bench blasting design for desired fragmentation based on digital image processing technique and Kuz-Ram model.* Int. Symp. on Rock Fragmentation by Blasting, Spain, P. 341-348.

48 Environmental Protection Agency (EPA), 2000. *Carbon dioxide emissions from the generation of electric power in the United States.* Department of Energy, Washington, USA.

49 Experian Catalist Group, 2012. *Fuel Price Report*. In: www.catalist.com.

50 Fordham P., 2004. *Mining company performance improvement programs and results-summary of benchmarking stu*dy. Plant Operators' Forum 2004.

51 Fortescue Metals Groups Ltd, 2010. *Analyses of diesel use for mine haul and transport operations*. The Depart. of Res., Energ. and Tour., Australian Government, Australia.

52 Gillot, L., 2005. *Pit-to-Plant optimisation at Morila Gold Mine*. Morila Gold Mine, Mali, West Africa.

53 Goodwin, C., et al., 2008. *Optimization opportunities in mining, metal and mineral processing*. Annual Reviews in Control, 32, 17–32.

54 Grouhel, P., 1991. *An explosive selection technique for optimizing blasting at open cut mining operations.* M.Sc. Thesis, University of Queensland.

55 Guanaco Mining Company, 2010. *Guanaco mine re-opening project*. Feasibility study report, Chile, v15, p.1-9.

56 Hart, S., et al., 2001. *Optimisation of the Cadia Hill SAG mill circuit*.

57 Hartman, H.L., 1992. *SME Mining Engineering Handbook*. Society of mining, metallurgy and exploration (SME), the 2nd ed., v1, P. 764-782.

58 Hartman, H.L., 1992. *SME, Materials Handling: Loading and Hauling*. (SME), the 2nd ed., v1, P. 2011-2017.

59 HUCKA, V. and DAS, B., 1975. *Laboratory investigation of penetration properties of the complete coal series.* Int. J. Rock Mech. Min. Sci. Geomech., vol. 12, P. 213–217.

60 IBRD, The International Bank for Reconstruction and Development, 2012. *Global Commodity Markets, Review and price forecast*. USA.

61 Imrie, C., 2001. *Ore Flow Optimization – Mine to Mill*. Hatch.

62 Indexmundi, 2012. *Commodity Price Indices*, http://www.indexmundi.com/ commodities/?commodity.

63 Kanchibotla, S., et al., 1999. *Exploring the effect of blast design on SAG mill throughput at KCGM.* Julius Kruttschnitt Mineral Research Centre.

64 Karanam, U.M. and Misra, B., 1998. *Principles of rock drilling*. Rotterdam, P. 111-144.

65 Kemeny, J., et al., 2001. *Improvements in blast fragmentation models using digital image processing.* Explo 2001, Inter. Conf., Australia, 28-31

66 Kennedy, B.A., 1990. Surface Mining. SME, Avondale, AZ, U.S.A., 2e, P. 400-405.

67 King, R.P., 2000. *Technical notes on jaw crushers*.

68 Kirmanli, C. and Ercelebi, S.G., 2009. *An expert system for hydraulic excavator and truck selection in surface mining.* The Southern Afr. Inst. of Min. and Metal., P. 727-738.

69 Kleemann, MOBICAT. *MC125Z Crusher technical catalog*. www.kleemann.info/de

70 Kuznetsov, V.M., 1973. *The mean diameter of the fragments formed by blasting rock*. Soviet Mining Science 9(2). P. 144-148.

71 Lane, K.F., 1964. *Choosing the optimum cut-off grades*. Colorado School Mines Q59, 811–824.

72 Lane, K.F., 1988. *The economic definition of Ore-Cut-Off Grades in theory and Practice*. Mining Journal Books Limited, London.

73 Lane, G.S., and Siddall, G.B., 2002. *SAG milling in Australia - focus on the future, metallurgical plant design and operating strategies*. AusIMM.

74 Lilly, P.A., 1986. *An empirical method of assessing rock mass blastability*. The Aus. IMM-IE Aust. Newman Combine Group, Large Open-pit Mining Conf., P. 89–92.

75 Lilly, D., 2007. *A statistical approach to integrating blasting into the mining process.* Oxford Business & Economics Conference.

76 Lilly, D., 2008. *The enterprise solution to integrating blasting into the mining process*. Oxford Business and Economics Conference.

77 London Metal Exchange, http://www.lme.com

78 Lotter, N. and Laplante, A., 2007a. *The campaign survey model-A case study at Raglan mine.* Minerals Engineering 20, pp. 480-486.

79 Lotter, N. and Laplante, A., 2007b. *Statistical benchmark surveying of production concentrators.* Minerals Engineering 20, pp. 793-801

80 Lowrie L., 2002. *SME Mining Reference Handbook*., P. 14.

81 MacKenzie, S., 1966. *Cost of explosives do you evaluate it properly?* Mining Congress Journal. American Mining Congress. P, 32–41.

82 Mackenzie, S., 1967. *Optimum blasting*. In: Annual Minnesota Mining Symposium, Duluth, MN, P. 181-188.

83 McGraw-Hill Companies, 2004. *Soil and Earthwork Formulas*. In Civil Engineering Formulas, CH. 8, P. 258-282.

84 Mine Safety and Health Administration, US Department of Labor, 2008, Washington, D.C., http://en.wikipedia.org/wiki/Mine_Safety_and_Health_Administration

85 Minnitt, R.C.A., 2003. *Cut-off grade determination for the maximum value of a small Wits-type gold mining operation.* Proceedings of the 31st Interna- tional Symposium on Application of Computers and Operations Research in the Minerals Industries (APCOM), Cape Town, South Africa.

86 Mintek mineralogists, 2007. *Looking into the ore*. International Mining, pp17-23.

87 Moon, C. J., et al., 2006. *Introduction to mineral exploration*. Hand book, Blackwell.

88 Morrell, S., and Morrison, R., 1996. *AG and SAG mill circuit selection and design by simulation.* International Conference on Autogenous and Semiautogenous Grinding.

89 Morell S., 1999. I*ncreasing mine-site profitability through optimisation of mining and processing operations.* JKMRC.

90 Technology, 2, p. 769-790, Vancouver, Canada.

91 Mosteller F. and Rourke K., 1973. *Sturdy Statistics*. Reading, MA: Addison-Wesley.

92 Muduli, P. and Yegulalp, T., 1999. *Modeling truck-shovel systems as closed queuing network with multiple job classes*. Inter. trans. in operational research, vol. 3, P. 89-98.

93 Ndibalema, A., 2008. *Capturing economic benefits from blasting*. The S. African Ins. of Min. and Metal., Geita gold mine, Tanzania, P.97-112.

94 Nelson, G., 2005. Application of odor Sensors to ore sorting and mill feed control. Final report, University of Utah Mining Engineering Department.

95 Nienhaus, K. and Fiona, 2009. *Automation and process monitoring in the mining industry by infrared sensor technology and industrial image processing.* Gluckauf mining reporter, G1-2009, 28-32

96 Noetstaller R., 1988. *Industrial Minerals: a technical review*. The World Bank, Washington.

97 Odell, C. J. and Scoble, M., 2005. *Tools for the integration of sustainability into the mine design process.* Proceedings of the 32nd. International symposium on the application of computers and operations research in the mineral industry (APCOM), 125-131.

98 Osanloo, M. and Ataei, M., 2003. *Using equivalent grade factors to find the optimum cut-off grades of multiple metal deposits.* Minerals Engineering 16, pp. 771-776

99 Osanloo, M., Rashidinejad, F. And Rezai, B, 2008. *Incorporating environmental issues into optimum cut-off grades modeling at porphyry copper deposits*. Resources Policy, 33, 222- 229

100 Osborne, S. and Maw, L., 2005. *Considering geology in drill and blast operations to optimize ore body value.* Orica Mining Services, Macraes, New Zealand.

101 Ouchterlony, F., 2005. *The Swebrec function: linking fragmentation by blasting and crushing*. Mining Technology (Trans. Inst. Min. Metall.), Vol.114, pp. 29-44.

102 Ozkahraman, T., 1994. *Critical evaluation of blast design parameters for discontinuous rock by slab blasting.* Ph.D. thesis, Middle East Technical University, Ankara.

103 Ozkahraman, T., 2006. *Fragmentation assessment and design of blast pattern at Goltas Limestone Quarry, Turkey*. Inter. J. of Rock Mechanics & Mining Sciences 43, 628–633.

104 Palangio, C., 1999. *Case Studies using the WipFrag Image Analysis System.* 6th Inter. Symp. for rock fragmentation by blasting, Johannesburg, South Africa , p. 117-120.

105 Putland, B., 2005. *An overview of single stage Autogenous and Semiautogenous grinding mills*. Orway Mineral Consultants, Australia.

106 Ramirez-Rodriguez, G.D. and Rozgonyi, T.G., 2004. *Evaluating the impact of environmental considerations in open pit mine design and planning.* (SWEMP), Antalya, Turkey.

107 Rashidinejad, F., et al., 2008. *An environmental oriented model for optimum cut-off grades in open pit mining projects to minimize acid mine drainage*. Int. J. Environ. Sci. Tech., 5 (2), 183-194

108 Rashidinejad, F., et al., 2009. *Cut-off grades optimization with Environmental management; a case study: Sungun copper project*. I. J. Eng. Sci., V19, P.1-13.

109 Rodgers, A., 1999. *Measurement Technology in Mining*. The Ensign-Bickford Company Simsbury, Connecticut, USA

110 Ross, S.A., et al., 2002. Corporate Finance. New York: McGraw-Hill.

111 Sahoo, L.K., et al., 2010. *Energy performance of dump trucks in opencast mine*. Inst. of techn., Bombay, India, P. 1-8.

112 Schivley, G.P., 1994. *Predicting rotary drill performance*. Proceedings of the 20th Conference on Explosive and Blasting Techniques, Int-Soc. Expl. Engineers, Inc., Cleveland, OHIO.

113 Schneider, C. L., and King, R. P., 1995, *A comprehensive simulation of an industrial comminution circuit treating taconite*. International Mineral Processing Congress, San Francisco, CA, October

114 Sgurev, V., et al., 2003. *An automated system for real-time control of the industrial truck haulage in open-pit mines*. Inst. of industrial cybernetics and robotics, Bulgaria.

115 Sharmaa, G., et al., 2009. Bayesian statistics and production reliability assessments for mining operations. Int. J. of min., reclam. and Env., Vol. 23, pp. 180-205.

116 Siddiqui, F.I., et al., 2009. *Measurement of size distribution of blasted rock using digital image processing*. Eng. Sci., Vl. 20, p. 81-93.

117 Simon G., 2004. *The normal distribution.* Documents prepared for use in course B01.1305, Stern School of Business, New York University, USA.

118 Srajer, V., et al., 1989. *Selection of loading and hauling equipment: user practices*. CANMET, Surface Mining Laboratories, Alberta, Canada.

119 Starkey, J.H., 2003. *Accurate, economical grinding design using SPI and Bond.* IMPC XXII Conference, Cape-town.

120 Strohmayr, S. and Valery, W., 2001. *SAG mill circuit optimization at Ernest Henry mining*. Inter. Conf. on AG and SAG Grinding Tech., Vancouver, Canada, V3, p.11-42.

121 Sylwestrzak, L., 2009. *Galvanox-tm An opportunity for existing copper flotation concentrators to improve overall project economics*. Hydro-copper 2009, Chile, P. 17-25.

122 Teale, R., 1965. *The concept of specific energy in rock drilling*. Int. J. Rock Mech. Min. Sci., P. 57-73.

123 Thwaites, P., 2007. *Process Control in metallurgical plants-From an Xstrata1 perspective.* Annual Reviews in Control 31, pp221-239.

124 Tilton J.E., et al., 1988. *World Mineral Exploration*. Resources for the Future, Washington.

125 Tolouei.R., 2009. *Vehicle mass as a determinant of fuel consumption and secondary safety performance.* Transport research Part D.

126 VanDelinder, R., et al., 2004. *Control of Blasting for Ore Blending and Autogenous Mill Performance at Hibbing Taconite Company-Preliminary Findings.* Plant Operators' Forum 2004.

127 Vensim reference manual, 2010. Ventana Systems, Inc. Revision data.

128 Wellmer F.W., 1998. *Problems related to cut-off levels*. In Statistical Evaluations in Exploration for Mineral Deposits, 151–159. Springer-Verlag, Berlin.

129 Wheeler, A. J. and Rodrigues, R. L., 2002. *Cut-off grade analysis at Fazenda Brasileiro: mine planning for declining gold prices*. Mining Tec., IMM Transactions section A, (1), 35-46.

130 Wikimedia Commons, http://commons.wikimedia.org

131 Willis, B. A., 1988. *Enhancement of Mineral Liberation.* Proceedings of XV International Minerals Processing Congress. P, 293-297

132 Workman, L. and Eloranta, J., 2002. *The effects of blasting on crushing and grinding efficiency and energy consumption*. Calder & Workman Inc.

133 Workman, L. and Eloranta, J., 2004. *The Effects of Blasting on Crushing and Grinding Efficiency and Energy Consumption*. Calder & Workman Inc.

134 www.business.highbeam.com/industry-reports/equipment/mining-machinery

135 www.catrental.net.au/Customer%20Support/Documents/SWELL

136 www.diracdelta.co.uk/science/source/s/p/specific%20fuel%20consumption/source.html

137 www.en.wikipedia.org/wiki/Brake_specific_fuel_consumption.

138 www.energy.eu. *Europe's Energy portal*, 2012.

139 www.finance.thinkanddone.com/profitability-index.html

140 www.greatwallmill.com

141 www.indexmundi.com/commodities/?commodity=copper&months

142 www.iseesystems.com/softwares/Education/StellaSoftware.aspx

143 www.solveitsoftware.com/mining/mining-simulation

144 www.theaa.com/public_affairs

Appendix 1: Assignment of the natural parameters inter-acting the integrated optimization

Ore deposit characterization and natural parameters

While the operational parameters, for each of mining or processing operations, tend to affect particularly its own stage, their outputs may have great effects on the other subsequent processes. On the contrary of this, the ore natural parameters tend to affect all the operations with, considerably, different results and influences on the concerned operation and the downstream other one(s).

In the followings, the characterized and effective ore deposit natural parameters, which will have considerable effects on the mining and processing activities, will be assigned and focused. This will be done in order to be linked to the cause and effect investigation for the inter-related mining and processing operations in the way to the integrated optimization.

Mineral liberation grain size

Mineral liberation grain size is considered the most important natural parameter. Depending on this parameter and on the resultants during achieving it, many other operational parameters belong to ore grinding and mineral concentration should be adjusted and adapted.

Realization of this parameter (liberation size) is the main target for milling and its precedent crushing process. But, as it is verified before, milling is considered the most critical process all over the other operations, due to its high consumption of electrical energy. The importance for this ore natural parameter is that its product assigns the real output tonnage and recovery of the valuable mineral products, whither they will be transferred to the next separation technical process or transported directly to the market.

Any final milled product should, practically, include the following three types of grain size, as shown in Figure 88, where the red parts denote the valuable mineral and the gray parts denote the gangue material:

➢ Grain size (A): the proportion of the ore that contains meddlings of the valuable minerals engaged with the other gangue materials, (not liberated).

➢ Grain size (B): the proportion of the ore that contains the valuable minerals with quite liberation but are also over milled to more fineness than required, (over milled).

➢ Grain size (C): the proportion of the ore that contains the valuable minerals with quite liberation to its required optimum size for the subsequent concentration process or the market handling.

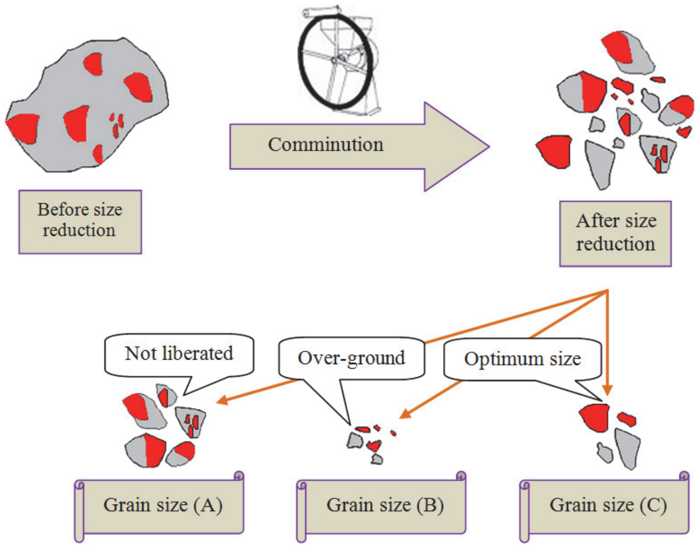


Fig. 88: Liberation size and the final ground product judgment.

The grain size distribution is considered here an important characterization for the milled final product. The main target for an optimized grinding, with the mill throughput being constant, is to increase the grain size type (C) to be maximum and to minimize the other grain size types (A) and (B) to be minimum.

The reason for adopting this strategy is as follows:

If grinding is designed to produce more fineness than required, the final product grain-size will contain a high proportion from the grain size (B) declaring that parts of the ore are over-milled. This is, of course, consuming more costs for a wasted energy.

On the other hand, if grinding is not sufficiently adjusted to produce the required fineness, the final product grain-size will contain a high proportion from the grain size (A) declaring that parts of the ore are not completely exposed. This causes a considerable reduction in the final product recovery by transferring an amount of the valuable mineral, engaged with the gangue, to tailings.

Rock texture and ore hardness

Hardness is considered an important natural factor that characterizing the different mined and processed ore deposits. This is owing to its direct inter-relation with the size reduction operations.

Regarding the ore fragments size reduction, hardness is considered a very active natural parameter inter-acts, especially, in grinding stage, while in the primary crushing stage, the internal fractures property acts as the most predominant.

The feeding and operating fragmentation within primary crushing is relatively much coarser. Hence, the grain hardness parameter, which belongs to the type, the interlocking matrix and the crystalline structure of the ore grains, will be of a little impact compared with that introduced by joints and fractures. As the grain size is decreased in the direction of grinding, the hardness becomes of a more and more importance until being the most prevailing natural parameter within the secondary grinding.

The difference in the hardness between ore and gangue minerals is an essential natural factor, required to be investigated, in order to be able to separate these interlocked and cemented materials. The interference between these linked materials always takes different and, may be complicated forms, Figure 89, [120].

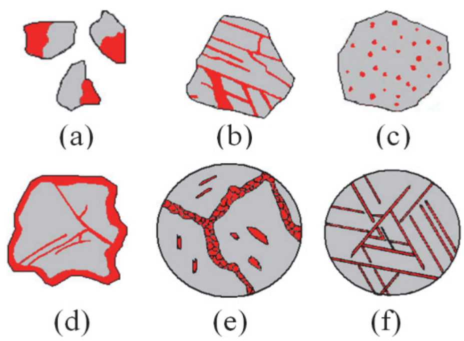


Fig. 89: Scheme for the different valuable and gangue minerals interference [120].

In Figure 91, the red color denotes the valuable mineral, while the gray denotes the gangue materials. Also a) Granular texture, (b) veined texture, (c) disseminated texture (d) supergenic-coated texture (e) exsolved texture (f) blades texture.

In the figure, consider, for example, the interlocking type (a) and assume that: H_A is less than H_B, where H_A is the hardness of the red portion (A) and H_B is the hardness of the gray portion (B).

Then the transverse rupture strength σ, which is considered to be related to the yielding or plastic deformation of ore particles, hence related to the particle hardness, can be expressed as [11]:

$$\sigma = \sigma_0 + \sqrt{C\frac{f^{2/3}}{d}} \quad (101)$$

Where: σ_0 yield strength of (A) when there is no (B), f (B) volume fraction within the particle; d grain diameter; and C material constant.

Considering the micro-structural factors, the overall particle hardness will increase with:

- Increasing of the harder portion, (f) and
- Decreasing of the particle size, (d)

When the volume fraction of (B) is fixed, then the yield strength (could be a measure of hardness) is proportional to $d^{-0.5}$. Hence, the ore particles tend to be harder with decreasing their size.

The interference forms between the linked materials and the crystallographic grain size could be recognized by different techniques, as microscopic investigation of the rock thin-sections, X-ray diffraction, Electron probe micro-analyzers and Ultraviolet fluorescence microscopy [116].

Identification of the hardness of the ROM real feeding mixture to the different size-reduction stages plays a great role in predicting of the prerequisite equipment designating electric power and the final total energy consumption of the processing plant.

Rock mechanical characteristics

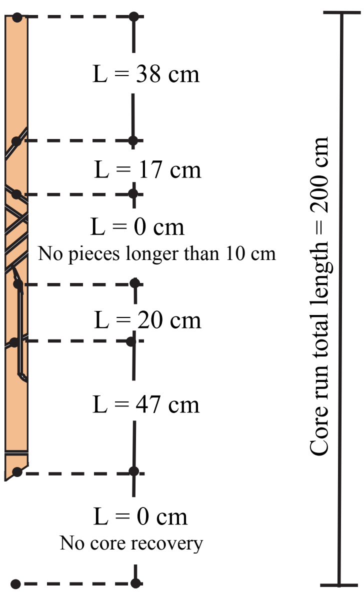
Materials vary widely in the natural properties, even within the same basic groups. Selection of the most appropriate mining strategy, machinery, crushers… is greatly influenced by the precise nature of the material to be mined.

Rock mass classification systems constitute an integral part of empirical mine design and planning. They are traditionally used to group areas of similar geo-mechanical characteristics, to provide guidelines for stability conditions, equipments selection and performance, fracturing, blasting and other energy consumption prediction.

The primary objective of all classification systems is to quantify the actual properties of the rock mass to investigate how the different rocks can influence the behavior of the mining operations.

Rock Quality Designation (RQD)

The Rock Quality Designation (RQD) is a drilling core recovery index and defined as the percentage of the total length of intact core greater than 10 cm in length, divided by the total length of the core run [32]. Figure 90 shows simple procedure example for determining RQD.



$$RQD = \frac{100 * \{\sum (\text{Length of core pieces} > 10\text{ cm})\}}{\text{Total core run length}}$$

$$RQD = \frac{100 * (38+17+20+43)}{200}$$

RQD = 59 % (FAIR)

RQD	R. classification
<25 %	Very poor
25-50%	Poor
50-75%	Fair
75-90%	Good
90-100%	Very good

Fig. 90: Procedure example for determining RQD [32].

The length of core used for the calculation is measured from fracture to fracture along the axis of the core. In determination of the RQD, the shear zones greater than 1m must be classified separately. Normally, RQD is calculated over individual core runs, usually 1.5 m long.

Intact lengths of core only consider joints or other natural discontinuities, so drill breaks should be ignored, otherwise the resulting RQD will be underestimated.

The main drawbacks to RQD are that it is sensitive to the direction of measurement, and it is insensitive to changes of joint spacing, if the spacing is over 1m. The main use of RQD is to provide a warning that the rock mass is probably of low quality.

Rock Mass Rating (RMR)

The Mining Rock Mass Rating (RMR) classification system [13], as an empirical system, was developed for characterizing and providing a design tool for the rock mass.

The system has a better understanding of the importance of the different parameters by adding of five different ratings, Table (Ap1-1), to give the final rating value of RMR. These parameters are:

- IRS (intact rock strength "unconfined uniaxial compressive strength");
- RQD (rock quality designation);
- Joint frequency (number of natural occurring fractures or discontinuities per length);
- Joint conditions (joint infill, persistence, and roughness); and
- Ground water conditions (water pressure)

In fact, RMR classification system is based on a rating of three principal rock property groups. These are the rock strength, the discontinuities frictional properties and the geometry of intact blocks of rock defined by the discontinuities.

The classification involves the assignment to the rock mass of an in situ rating based on measurable geological parameters. The range of 8 to 100, divided into five classes, is used to cover all variations in jointed rock masses from very poor to very good.

It is essential that, the classification data are made available at an early stage, so that the correct decisions are made on mining method, layout, machinery, etc. Using the RMR, the rock mass can be described in numbers, which define mainly the strength of the material. Excavation stability, fragmentation, economic considerations, machinery, etc., could be related to these numbers.

Spacing of fractures and joints (Continuity)

Spacing is the measurement of all the discontinuities and partings, and does not include cemented features. Cemented features affect the IRS and are included in its determination.

A joint is obvious, which is continuous if its length is greater than the breakage width or if it intersects another joint. Thus, joints define blocks of rock. As the rock mass includes both continuous (joints) and discontinuous (fractures) features, the continuity must be estimated to give the joint spacing, considering the rock block size.

Table (Ap1-1): Mining rock mass rating (RMR) [13].

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS									
Parameter			**Range of values**						
1	**Strength of intact rock material**	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	Uniaxial Compres. test is preferred		
		Uniaxial comp. strength	>250 MPa	100-250 MPa	50 –100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		**15**	**12**	**7**	**4**	**2**	**1**	**0**
2	**Drill core Quality RQD**		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		**20**	**17**	**13**	**8**	**3**		
3	**Spacing of discontinuities**		> 2 m	2. - 0.6 m	0.6 - 0.2 m	200- 60 mm	< 60 mm		
	Rating		**20**	**15**	**10**	**8**	**5**		
4*	**Condition of discontinuities**		Very rough surfaces Not continuous No separation Un-weathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered	Slightly rough surfaces Separation < 1 mm Highly weathered	Slickensided surfaces Gouge < 5 mm thick Separation 1-5 mm Continuous	Soft gouge >5 mm thick Separation > 5 mm Continuous		
	Rating		**30**	**25**	**20**	**10**	**0**		
5	**Ground water**	Inflow/10 m (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water pres.) / (Major principal σ)	0	< 0.1	0.1 - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		**15**	**10**	**7**	**4**	**0**		

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS						
Strike and dip orientations		Very favorable	Favorable	Fair	Unfavorable	Very Unfavorable
Ratings	Tunnels & mines	0	-2	-5	-10	-12
	Slopes	0	-5	-25	-50	-60

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS					
Rating	**100 – 81**	**80 - 61**	**60 - 41**	**40- 21**	**< 21**
Class number	**I**	**II**	**III**	**IV**	**V**
Description	**Very good rock**	**Good rock**	**Fair rock**	**Poor rock**	**Very poor rock**

D. MEANING OF ROCK CLASSES					
Class number	**I**	**II**	**III**	**IV**	**V**
Cohesion of rock mass (kPa)	**> 400**	**300 - 400**	**200 - 300**	**100 - 200**	**< 100**
Friction angle of rock (deg)	**> 45**	**35 - 45**	**25 – 35**	**15 - 25**	**< 15**

E*. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions					
Discontinuity length Rating	< 1 m 6	1 - 3 m 4	3 - 10 m 2	10 - 20 m 1	> 20 m 0
Separation (aperture) Rating	None 6	< 0.1 mm 5	0.1 - 1.0 mm 4	1 - 5 mm 1	> 5 mm 0
Roughness Rating	Very rough 6	Rough 5	Slightly rough 3	Smooth 1	Slickensided 0
Infilling (gouge) Rating	None 6	Hard filling <5mm 4	Hard filling >5mm 2	Soft filling <5mm 2	Soft filling >5mm 0
Weathering Ratings	Unweathered 6	Slightly weathered 5	Moderately weathered 3	Highly weathered 1	Decomposed 0

* Rating of item No. 4 = ∑for items in E

Joint Condition and Water

Joint condition is an assessment of the frictional properties of the joints (not the fractures) and is based on the surface properties, alteration zones, filling, etc. However the effect of water is in a separate section, the joint condition is allowing for water inflow to have a great sensitivity.

Rock Mass Strengths (RMS) and stability

Rock mass strength is the strength of the inhomogeneous mass, in which the excavation or breakage will be occurred. It is possible to use the values of (RMR) to determine an empirical rock-mass strength (RMS) in Mega-Pascal (MPa). Then RQD, joint spacing, and condition will act as a function in (RMS) determination, which cannot be higher than the average intact rock strength (IRS) of that zone.

Rock mass strength, as most other natural parameters, is scale dependent. For example, failure in a small slope in a hard, competent, ore would be controlled by discontinuities, not by the inherent strength of the rock mass. However, if the slope height was increased and zones of varying strength material were included, then failure could conceivably pass through the intact rock.

Internal Fractures

Internal fractures of the ore blocks are the results of following of the stressing forces to the naturally existed week paths within the ore body structure. These stressing forces begin during blasting, where macro-fractures are induced to the fragmented blocks, Figure 91 [44]. This fracture type has a great importance to the muck-piles fragment size distribution and upper size.

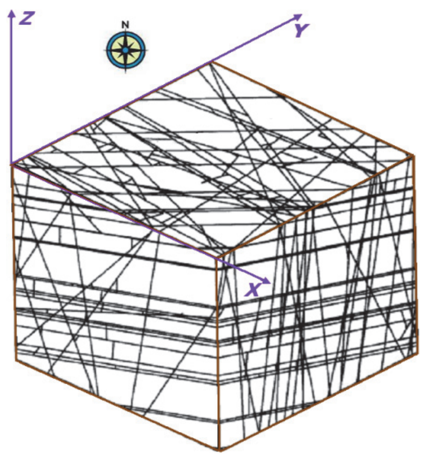


Fig. 91: Fractures modeling for a geological block model [44].

Micro-fractures develop around mineral grains and their densities can be indicated by a microscope from the rock thin sections. There are three groups of microscopic-cracks:

- Small cracks: occur only along a short length of the boundary between mineral grains, or go into a single grain without cutting through it.
- Boundary cracks: can be traced along the boundaries of several grains.
- Grain cracks: cut entirely through one or more grains.

These micro internal cracks, especially the last two categories, are so important in crushing and they affect its production and bridging times and still present also in grinding feed. The main effect of internal fractures is to soften the fragments, making them easier to break, Figure 92. This has benefits to productivity, energy expenditure, and wear of consumable parts.

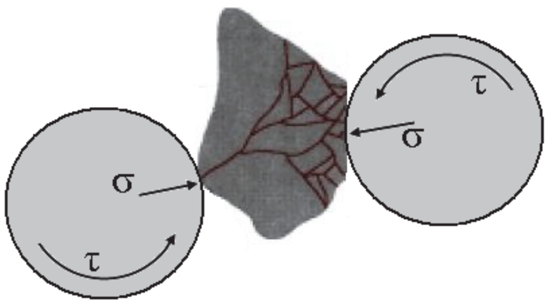


Fig. 92: Induced cracks through the internal fractures due to different acting force.

Friability and Brittleness

Brittleness is a property of materials that rupture with a little or no plastic flow and is considered one of the important mechanical properties of rocks, as most of the rocks show brittle fracture when failed under loads. Brittleness can be taken as a term describes the rocks fracture-type [33] and can be defined by the ratio of elastic strain to plastic strain at fracturing. The higher this ratio, the higher is the brittleness of the fracture and the lower its ductility [4, 5].

With higher brittleness, the following facts are observed [59]:

- formation of fines
- higher ratio of compressive to tensile strength
- higher resilience
- higher angle of internal friction
- formation of cracks in indentation.

The Brittleness index (BI) has significant correlations with penetration rate of percussive drills and the specific drilling energy. The brittleness index, defined as the area under the line of the compressive strength versus tensile strength, can be formulated as follows:

$$BI = \frac{\sigma_C \times \sigma_T}{2} \qquad (102)$$

Where: σ_C uniaxial compressive strength of rock, (MPa) and σ_T tensile strength of rock (MPa).

Friability is the material ability to produce fines when exposed to pressure; and its value is measured from the brittleness test. The brittleness test, Figure 93, gives a value for rock resistance to crushing due to repeated weight-drop impacts [33].

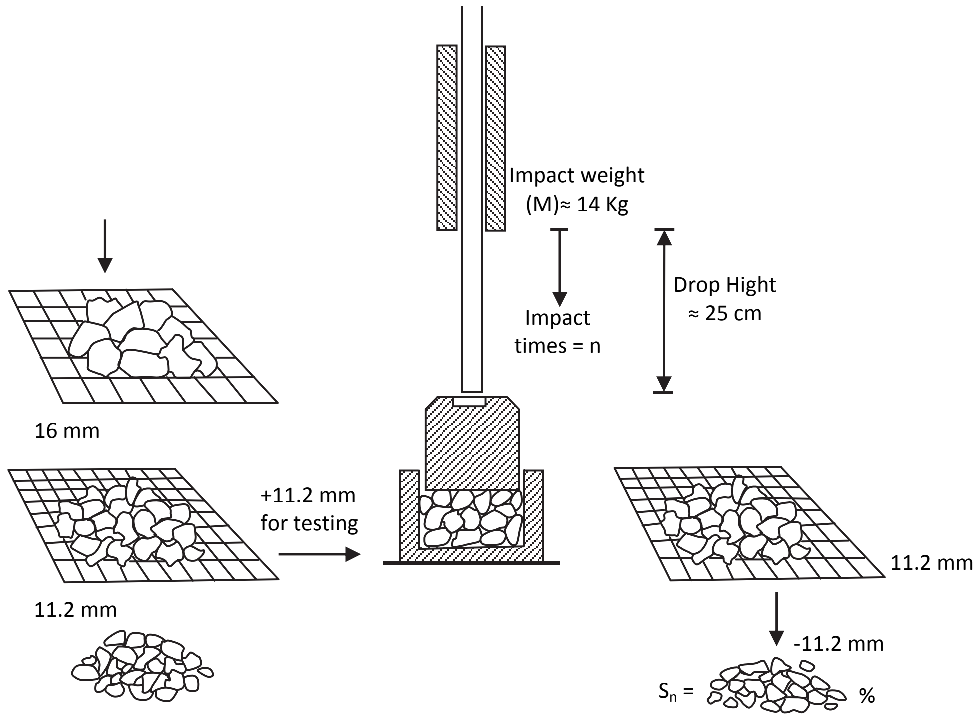


Fig. 93: The brittleness test [33].

The friability value equals the percentage of undersized material, which passes through the (11.2 mm) size after crushing by a number of the drop weight of an energy E_{drop}:

$$E_{drop} = M_{drop} \times g \times H_{drop} \qquad (103)$$

Where: M_{drop} mass of the weight drop on the sample, g gravitational force, and H_{drop} height, from where the weight falls on the sample.

In general, the drop weight tests of minimum three or four parallel tests are to be carried out and the mean value is considered for determination of friability value.

The work index depends on the strength of the interlocking matrix and the micro- structure of the crystals. In brittleness test, the weight drops number is dependent on the tested material tensile strength; and as the tensile strength of rocks correlates better with work index than with compressive strength and point load index, the brittleness test can be an indicator to the work index and grind-ability characteristics of rocks.

Ore Toughness

Rock fracture toughness, as a natural parameter in fracture mechanics, represents the ability of a material to resist the propagation of cracks, and thus it can be a measure of the energy required to create new surface areas in the material. It is considered also as an inherent property of the material and it is not affected by the configuration of the specimen.

The main use of this property is for classification of intact rock with respect to its resistance to crack propagation as seen in rock cutting. Fracture toughness is comparatively more difficult to obtain than standard laboratory rock properties such as compressive and tensile strengths.

The toughness index is defined as the strain energy stored in a unit volume of rock just before the failure. Therefore, it is referred to as the amount of energy required to cause breakage. The ratio of hardness to toughness can be also proposed as an index of brittleness [59].

Mineral Content

The mineral content is a very important natural parameter and is needed to be continually investigated, especially in the case of the valuable and metallic mineral deposits, in order to realize a profitable dealing on mining and processing.

The mineral content factor should be always contoured, traced and controlled during the various mining and processing stages, beginning during drilling and ending with the product marketable form.

Appendix 2: Criteria for the suitable practicable fragmentation range, (Transports strategy B)

Table (Ap2-1): Fragmentation size model runs, results and criteria for choice of the suitable practicable range (Using transports strategy B).

Item/Run	Unit	1	2	3	4	5	6	7	8	9	10	11
Mean fragment size	m	**0.15**	**0.20**	**0.25**	**0.30**	**0.35**	**0.40**	**0.45**	**0.50**	**0.55**	**0.60**	**0.65**
Spec. explosive energy	kg/m^3	3.23	2.05 1.5	1.44	1.08	0.85	0.69	0.57	0.48 0.5	0.41	0.36	0.32
Spec. drilling & blasting cost	$/t	0.845	0.586	0.452	0.372	0.321	0.286	0.260	0.241	0.226	0.214	0.205
Lumpiness size	m	0.23	0.31	0.40	0.49	0.59	0.70	0.80	0.91	~~1.03~~	~~1.15~~	~~1.28~~
Bucket filling factor	Dmnl	0.92	0.87	0.825	0.76	0.69	0.61	0.60	0.59	0.58	0.56	0.55
Bulk density	t/m^3	2.25	2.14	2.03	1.87	1.69	1.53	1.50	1.47	1.41	1.37	1.34
Ore delivery rate	Ton/h	~~3568.17~~	~~3203.32~~	2859.24	2421.93	1889.53	1522.62	1449.53	1342.63	1268.21	1146.49	1044.82
Specific fuel consumption	kg/t	0.421	0.435	0.451	0.479	0.518	0.572	0.585	0.593	0.611	0.628	0.645
Fleet availability	%	92	92	92	91	91	90	89	89	89	89	88
Fleet utilization	%	~~105~~	99	93	86	78	69	67	66	65	64	62
Transport efficiency	%	97	91	86	78	71	62	60	59	58	57	55
Loading and hauling spec. cost	$/t	1.442	1.421	1.395	1.352	1.291	1.257	1.273	1.282	1.302	1.323	1.341
Total mining specific cost	$/t	2.287	2.006	1.847	1.725	1.613	1.543	1.533	1.523	1.529	1.537	1.546

Appendix 3: Annual tonnage weighted parameters for each mixing scenario, (Output of the Excel calculation tool)

Table (Ap3-1): Annual tonnage weighted averages of the main input parameters for the mixing scenarios (4-14) (Output of the Excel calculation tool)

Mix. Sc 4	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	123.1	123.1	123.1	123.1	123.1	123.1	123.1	123.1	122.4	118.9	118.9	118.9	118.9	118.9	118.9	114.7	95.0
Work Index (FG)	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.62	17.25	17.25	17.25	17.25	17.25	17.25	16.86	15.00
WI (CG)	15.50	15.50	15.50	15.50	15.50	15.50	15.50	15.50	15.62	16.25	16.25	16.25	16.25	16.25	16.25	15.86	14.00
WI (PC)	14.50	14.50	14.50	14.50	14.50	14.50	14.50	14.50	14.62	15.25	15.25	15.25	15.25	15.25	15.25	14.86	13.00
Liberation size	208	208	208	208	208	208	208	208	209	212	212	212	212	212	212	218	250
Distribution st. Dev.	53	53	53	53	53	53	53	53	52	47	47	47	47	47	47	46	40
Metal content	0.93	0.93	0.93	0.93	0.93	0.93	0.93	0.93	0.92	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.90
Final size	120	120	120	120	120	120	120	120	122	133	133	133	133	133	133	142	185
Penetration rate	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.3	29.7	29.7	29.7	29.7	29.7	29.7	31.5	40.0
Discontinuity	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.3	1.3	1.3	1.3	1.3	1.3	1.2	0.7
Mix. Sc 5	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Strength	124.9	124.9	124.9	124.9	124.9	124.9	124.9	124.9	124.9	124.9	124.9	121.7	121.5	121.5	101.4	95.0	95.0
Work Index (FG)	16.90	16.90	16.90	16.90	16.90	16.90	16.90	16.90	16.90	16.90	16.90	17.47	17.50	17.50	15.61	15.00	15.00
WI (CG)	15.90	15.90	15.90	15.90	15.90	15.90	15.90	15.90	15.90	15.90	15.90	16.47	16.50	16.50	14.61	14.00	14.00
WI (PC)	14.90	14.90	14.90	14.90	14.90	14.90	14.90	14.90	14.90	14.90	14.90	15.47	15.50	15.50	13.61	13.00	13.00
Liberation size	205	205	205	205	205	205	205	205	205	205	205	207	208	208	240	250	250
Distribution st. Dev.	53	53	53	53	53	53	53	53	53	53	53	48	48	48	42	40	40
Metal content	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.85	0.85	0.85	0.89	0.90	0.90
Final size	117	117	117	117	117	117	117	117	117	117	117	127	128	128	171	185	185
Penetration rate	29.1	29.1	29.1	29.1	29.1	29.1	29.1	29.1	29.1	29.1	29.1	28.5	28.5	28.5	37.2	40.0	40.0
Discontinuity	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.4	1.4	0.9	0.7	0.7
Mix. Sc 6	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Strength	126.7	126.7	126.7	126.7	126.7	126.7	126.7	126.7	126.7	126.7	126.7	126.7	116.4	108.1	95.0	95.0	95.0
Work Index (FG)	17.30	17.30	17.30	17.30	17.30	17.30	17.30	17.30	17.30	17.30	17.30	17.30	16.00	15.38	15.00	15.00	15.00
WI (CG)	16.30	16.30	16.30	16.30	16.30	16.30	16.30	16.30	16.30	16.30	16.30	16.30	15.00	14.38	14.00	14.00	14.00
WI (PC)	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30	14.00	13.38	13.00	13.00	13.00
Liberation size	201	201	201	201	201	201	201	201	201	201	201	201	218	231	250	250	250
Distribution st. Dev.	52	52	52	52	52	52	52	52	52	52	52	52	51	48	40	40	40
Metal content	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.93	0.94	0.90	0.90	0.90
Final size	114	114	114	114	114	114	114	114	114	114	114	114	134	153	185	185	185
Penetration rate	27.8	27.8	27.8	27.8	27.8	27.8	27.8	27.8	27.8	27.8	27.8	27.8	33.0	36.2	40.0	40.0	40.0
Discontinuity	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.1	0.9	0.7	0.7	0.7

Table (Ap3-1): Annual tonnage weighted averages of the main input parameters for the mixing scenarios (4-14) (Output of the Excel calculation tool), *Continued…..*

Mix. Sc 7	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	124.3	124.3	124.3	124.3	124.3	124.3	124.3	124.3	124.3	124.3	124.3	124.3	124.3	122.2	101.4	95.0	95.0
Work Index (FG)	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.37	15.61	15.00	15.00
WI (CG)	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.37	14.61	14.00	14.00
WI (PC)	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.37	13.61	13.00	13.00
Liberation size	205	205	205	205	205	205	205	205	205	205	205	205	205	207	240	250	250
Distribution st. Dev.	52	52	52	52	52	52	52	52	52	52	52	52	52	49	42	40	40
Metal content	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.86	0.89	0.90	0.90
Final size	118	118	118	118	118	118	118	118	118	118	118	118	118	125	171	185	185
Penetration rate	29.0	29.0	29.0	29.0	29.0	29.0	29.0	29.0	29.0	29.0	29.0	29.0	29.0	28.6	37.2	40.0	40.0
Discontinuity	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	0.9	0.7	0.7
Mix. Sc 8	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Strength	123.2	123.2	123.2	123.2	123.2	123.2	123.2	123.2	123.2	123.2	123.2	123.2	112.9	109.0	109.0	112.7	130.0
Work Index (FG)	17.20	17.20	17.20	17.20	17.20	17.20	17.20	17.20	17.20	17.20	17.20	17.20	15.90	15.40	15.40	15.51	16.00
WI (CG)	16.20	16.20	16.20	16.20	16.20	16.20	16.20	16.20	16.20	16.20	16.20	16.20	14.90	14.40	14.40	14.51	15.00
WI (PC)	15.20	15.20	15.20	15.20	15.20	15.20	15.20	15.20	15.20	15.20	15.20	15.20	13.90	13.40	13.40	13.51	14.00
Liberation size	206	206	206	206	206	206	206	206	206	206	206	206	223	230	230	225	200
Distribution st. Dev.	50	50	50	50	50	50	50	50	50	50	50	50	49	48	48	50	60
Metal content	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.88	0.92	0.94	0.94	0.95	1.00
Final size	122	122	122	122	122	122	122	122	122	122	122	122	143	151	151	142	100
Penetration rate	28.8	28.8	28.8	28.8	28.8	28.8	28.8	28.8	28.8	28.8	28.8	28.8	34.0	36.0	36.0	34.9	30.0
Discontinuity	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.0	0.9	0.9	1.0	1.2
Mix. Sc 9	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Strength	129.4	129.4	129.4	129.4	129.4	129.4	129.4	129.4	129.4	129.4	128.0	113.4	113.4	96.8	95.0	95.0	95.0
Work Index (FG)	17.55	17.55	17.55	17.55	17.55	17.55	17.55	17.55	17.55	17.55	17.37	15.53	15.53	15.05	15.00	15.00	15.00
WI (CG)	16.55	16.55	16.55	16.55	16.55	16.55	16.55	16.55	16.55	16.55	16.37	14.53	14.53	14.05	14.00	14.00	14.00
WI (PC)	15.55	15.55	15.55	15.55	15.55	15.55	15.55	15.55	15.55	15.55	15.37	13.53	13.53	13.05	13.00	13.00	13.00
Liberation size	197	197	197	197	197	197	197	197	197	197	199	224	224	247	250	250	250
Distribution st. Dev.	53	53	53	53	53	53	53	53	53	53	53	51	51	41	40	40	40
Metal content	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.89	0.95	0.95	0.91	0.90	0.90	0.90
Final size	108	108	108	108	108	108	108	108	108	108	111	140	140	181	185	185	185
Penetration rate	26.7	26.7	26.7	26.7	26.7	26.7	26.7	26.7	26.7	26.7	27.4	34.8	34.8	39.5	40.0	40.0	40.0
Discontinuity	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.0	1.0	0.7	0.7	0.7	0.7

Table (Ap3-1): Annual tonnage weighted averages of the main input parameters for the mixing scenarios (4-14) (Output of the Excel calculation tool), *Continued….*

Mix. Sc 10	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	124.0	124.0	124.0	124.0	124.0	124.0	124.0	124.0	124.0	123.3	120.2	120.2	120.2	120.2	120.0	95.0	95.0
Work Index (FG)	16.70	16.70	16.70	16.70	16.70	16.70	16.70	16.70	16.70	16.82	17.38	17.38	17.38	17.38	17.36	15.00	15.00
WI (CG)	15.70	15.70	15.70	15.70	15.70	15.70	15.70	15.70	15.70	15.82	16.38	16.38	16.38	16.38	16.36	14.00	14.00
WI (PC)	14.70	14.70	14.70	14.70	14.70	14.70	14.70	14.70	14.70	14.82	15.38	15.38	15.38	15.38	15.36	13.00	13.00
Liberation size	206	206	206	206	206	206	206	206	206	207	210	210	210	210	210	250	250
Distribution st. Dev.	53	53	53	53	53	53	53	53	53	52	47	47	47	47	47	40	40
Metal content	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.91	0.85	0.85	0.85	0.85	0.85	0.90	0.90
Final size	118	118	118	118	118	118	118	118	118	120	130	130	130	130	131	185	185
Penetration rate	29.8	29.8	29.8	29.8	29.8	29.8	29.8	29.8	29.8	29.6	29.1	29.1	29.1	29.1	29.2	40.0	40.0
Discontinuity	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	0.7	0.7

Mix. Sc 11	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	126.65	126.65	126.65	126.65	126.65	126.65	126.65	126.65	126.65	125.97	122.83	122.83	122.83	110.69	95.00	95.00	95.00
Work Index (FG)	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	17.07	17.63	17.63	17.63	16.48	15.00	15.00	15.00
WI (CG)	15.95	15.95	15.95	15.95	15.95	15.95	15.95	15.95	15.95	16.07	16.63	16.63	16.63	15.48	14.00	14.00	14.00
WI (PC)	14.95	14.95	14.95	14.95	14.95	14.95	14.95	14.95	14.95	15.07	15.63	15.63	15.63	14.48	13.00	13.00	13.00
Liberation size	202	202	202	202	202	202	202	202	202	203	205	205	205	225	250	250	250
Distribution st. Dev.	54	54	54	54	54	54	54	54	54	52	48	48	48	44	40	40	40
Metal content	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.92	0.90	0.85	0.85	0.85	0.87	0.90	0.90	0.90
Final size	112	112	112	112	112	112	112	112	112	114	125	125	125	151	185	185	185
Penetration rate	28.6	28.6	28.6	28.6	28.6	28.6	28.6	28.6	28.6	28.5	27.9	27.9	27.9	33.2	40.0	40.0	40.0
Discontinuity	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.4	1.4	1.4	1.1	0.7	0.7	0.7

Mix. Sc 12	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	133.8	133.8	133.8	133.8	133.8	133.8	133.8	133.8	116.4	112.5	112.5	112.5	112.5	108.1	95.0	95.0	95.0
Work Index (FG)	18.20	18.20	18.20	18.20	18.20	18.20	18.20	18.20	16.00	15.50	15.50	15.50	15.50	15.38	15.00	15.00	15.00
WI (CG)	17.20	17.20	17.20	17.20	17.20	17.20	17.20	17.20	15.00	14.50	14.50	14.50	14.50	14.38	14.00	14.00	14.00
WI (PC)	16.20	16.20	16.20	16.20	16.20	16.20	16.20	16.20	14.00	13.50	13.50	13.50	13.50	13.38	13.00	13.00	13.00
Liberation size	189	189	189	189	189	189	189	189	218	225	225	225	225	231	250	250	250
Distribution st. Dev.	53	53	53	53	53	53	53	53	51	50	50	50	50	48	40	40	40
Metal content	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.93	0.95	0.95	0.95	0.95	0.94	0.90	0.90	0.90
Final size	99	99	99	99	99	99	99	99	134	143	143	143	143	153	185	185	185
Penetration rate	24.2	24.2	24.2	24.2	24.2	24.2	24.2	24.2	33.0	35.0	35.0	35.0	35.0	36.2	40.0	40.0	40.0
Discontinuity	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.1	1.0	1.0	1.0	1.0	0.9	0.7	0.7	0.7

Table (Ap3-1): Annual tonnage weighted averages of the main input parameters for the mixing scenarios (4-14) (Output of the Excel calculation tool), *Continued…..*

Mix. Sc 13	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	126.6	126.6	126.6	126.6	126.6	126.6	126.6	123.4	121.5	121.5	121.5	121.5	121.5	121.5	101.4	95.0	95.0
Work Index (FG)	16.60	16.60	16.60	16.60	16.60	16.60	16.60	17.17	17.50	17.50	17.50	17.50	17.50	17.50	15.61	15.00	15.00
WI (CG)	15.60	15.60	15.60	15.60	15.60	15.60	15.60	16.17	16.50	16.50	16.50	16.50	16.50	16.50	14.61	14.00	14.00
WI (PC)	14.60	14.60	14.60	14.60	14.60	14.60	14.60	15.17	15.50	15.50	15.50	15.50	15.50	15.50	13.61	13.00	13.00
Liberation size	203	203	203	203	203	203	203	206	208	208	208	208	208	208	240	250	250
Distribution st. Dev.	55	55	55	55	55	55	55	50	48	48	48	48	48	48	42	40	40
Metal content	0.94	0.94	0.94	0.94	0.94	0.94	0.94	0.88	0.85	0.85	0.85	0.85	0.85	0.85	0.89	0.90	0.90
Final size	111	111	111	111	111	111	111	121	128	128	128	128	128	128	171	185	185
Penetration rate	29.4	29.4	29.4	29.4	29.4	29.4	29.4	28.8	28.5	28.5	28.5	28.5	28.5	28.5	37.2	40.0	40.0
Discontinuity	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.4	1.4	1.4	1.4	1.4	1.4	0.9	0.7	0.7

Mix. Sc 14	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Strength	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.4	120.3
Work Index (FG)	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77	16.77
WI (CG)	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77	15.77
WI (PC)	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77	14.77
Liberation size	211	211	211	211	211	211	211	211	211	211	211	211	211	211	211	211	211
Distribution st. Dev.	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50
Metal content	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
Final size	128	128	128	128	128	128	128	128	128	128	128	128	128	128	128	128	128
Penetration rate	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4
Discontinuity	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2

Appendix 4: Mass flow, economic and environmental output for the (*Avg-Method*) overall scenarios

Table (Ap4-1): Feeding rate (primary crushers) for each mixing scenario, (t/h), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1918	1917
2	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1918	1917
3	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1918	1917
4	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1918	1917
5	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1918	1917
6	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1918	1917
7	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1918	1917
8	1924	1920	1925	1916	1919	1921	1919	1920	1923	1918	1919	1927	1920	1917
9	1924	1921	1925	1917	1919	1921	1919	1920	1923	1918	1919	1912	1921	1917
10	1921	1923	1922	1919	1919	1921	1919	1920	1923	1918	1920	1908	1921	1917
11	1908	1923	1909	1919	1919	1921	1919	1920	1922	1920	1922	1908	1921	1917
12	1908	1923	1909	1919	1920	1921	1919	1920	1909	1920	1922	1908	1921	1917
13	1908	1921	1905	1919	1921	1912	1919	1910	1909	1920	1922	1908	1921	1917
14	1908	1901	1901	1919	1921	1907	1920	1907	1902	1920	1913	1907	1921	1917
15	1908	1901	1901	1919	1906	1901	1906	1907	1901	1919	1901	1901	1906	1917
16	1903	1901	1901	1915	1901	1901	1901	1909	1901	1901	1901	1901	1901	1917
17	1901	1901	1901	1901	1901	1901	1901	1916	1901	1901	1901	1901	1901	1917

Table (Ap4-2): Feeding rate (fine grinding) for each mixing scenario, (t/h), (Avg-Method)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
2	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
3	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
4	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
5	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
6	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
7	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
8	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2183	2181	2180
9	2182	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2179	2181	2180
10	2181	2181	2182	2180	2181	2181	2181	2181	2182	2180	2181	2178	2181	2180
11	2178	2181	2178	2180	2181	2181	2181	2181	2181	2181	2181	2178	2181	2180
12	2178	2181	2178	2180	2181	2181	2181	2181	2178	2181	2181	2178	2181	2180
13	2178	2181	2177	2180	2181	2179	2181	2178	2178	2181	2181	2178	2181	2180
14	2178	2175	2175	2180	2181	2177	2181	2177	2176	2181	2179	2177	2181	2180
15	2178	2175	2175	2180	2177	2175	2177	2177	2175	2181	2175	2175	2177	2180
16	2176	2175	2175	2179	2175	2175	2175	2178	2175	2175	2175	2175	2175	2180
17	2175	2175	2175	2175	2175	2175	2175	2180	2175	2175	2175	2175	2175	2180

Table (Ap4-3): Primary crushing by-pass for each mixing scenario, (t/h), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	339	342	337	346	344	341	343	343	339	345	343	335	345	346
2	339	342	337	346	344	341	343	343	339	345	343	335	345	346
3	339	342	337	346	344	341	343	343	339	345	343	335	345	346
4	339	342	337	346	344	341	343	343	339	345	343	335	345	346
5	339	342	337	346	344	341	343	343	339	345	343	335	345	346
6	339	342	337	346	344	341	343	343	339	345	343	335	345	346
7	339	342	337	346	344	341	343	343	339	345	343	335	345	346
8	339	342	337	346	344	341	343	343	339	345	343	335	343	346
9	339	342	337	346	344	341	343	343	339	345	343	350	342	346
10	342	340	340	344	344	341	343	343	339	345	343	354	342	346
11	355	340	353	344	344	341	343	343	341	343	341	354	342	346
12	355	340	353	344	342	341	343	343	353	343	341	354	342	346
13	355	341	357	344	342	350	343	352	353	343	341	354	342	346
14	355	362	362	344	342	356	342	355	361	343	350	356	342	346
15	355	362	362	344	357	362	357	355	362	343	362	362	357	346
16	360	362	362	347	362	362	362	354	362	362	362	362	362	346
17	362	362	362	362	362	362	362	346	362	362	362	362	362	346

Table (Ap4-4): Fine grinding by-pass for each mixing scenario, (t/h), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	81	81	80	82	82	81	82	82	81	82	82	80	82	82
2	81	81	80	82	82	81	82	82	81	82	82	80	82	82
3	81	81	80	82	82	81	82	82	81	82	82	80	82	82
4	81	81	80	82	82	81	82	82	81	82	82	80	82	82
5	81	81	80	82	82	81	82	82	81	82	82	80	82	82
6	81	81	80	82	82	81	82	82	81	82	82	80	82	82
7	81	81	80	82	82	81	82	82	81	82	82	80	82	82
8	81	81	80	82	82	81	82	82	81	82	82	80	82	82
9	81	81	80	82	82	81	82	82	81	82	82	84	82	82
10	81	81	81	82	82	81	82	82	81	82	82	85	82	82
11	85	81	84	82	82	81	82	82	81	82	81	85	82	82
12	85	81	84	82	82	81	82	82	84	82	81	85	82	82
13	85	81	86	82	82	84	82	84	84	82	81	85	82	82
14	85	87	87	82	82	85	82	85	87	82	84	85	82	82
15	85	87	87	82	86	87	86	85	87	82	87	87	86	82
16	86	87	87	83	87	87	87	85	87	87	87	87	87	82
17	87	87	87	87	87	87	87	82	87	87	87	87	87	82

Table (Ap4-5): Energy rate consumption (primary crushing) for each mixing scenario, (kW), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	753	730	756	713	726	739	729	736	748	719	728	769	716	721
2	753	730	756	713	726	739	729	736	748	719	728	769	716	721
3	753	730	756	713	726	739	729	736	748	719	728	769	716	721
4	753	730	756	713	726	739	729	736	748	719	728	769	716	721
5	753	730	756	713	726	739	729	736	748	719	728	769	716	721
6	753	730	756	713	726	739	729	736	748	719	728	769	716	721
7	753	730	756	713	726	739	729	736	748	719	728	769	716	721
8	753	730	756	713	726	739	729	736	748	719	728	769	735	721
9	753	733	756	717	726	739	729	736	748	719	728	696	745	721
10	739	754	743	737	726	739	729	736	748	723	732	680	745	721
11	678	754	681	737	726	739	729	736	742	741	755	680	745	721
12	678	754	681	737	744	739	729	736	685	741	755	680	745	721
13	678	749	672	737	745	696	729	692	685	741	755	680	745	721
14	678	662	662	737	745	675	741	676	664	741	712	675	745	721
15	678	662	662	737	682	662	682	676	662	740	662	662	682	721
16	667	662	662	724	662	662	662	680	662	662	662	662	662	721
17	662	662	662	662	662	662	662	697	662	662	662	662	662	721

Table (Ap4-6): Energy rate consumption (fine grinding) for each mixing scenario, (kW), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	20300	20503
2	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	20300	20503
3	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	20300	20503
4	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	20300	20503
5	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	20300	20503
6	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	20300	20503
7	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	20300	20503
8	21659	20795	21786	20173	20668	21163	20789	21036	21474	20421	20732	22282	21000	20503
9	21659	20942	21786	20321	20668	21163	20789	21036	21474	20421	20732	19569	21405	20503
10	21156	21716	21283	21094	20668	21163	20789	21036	21474	20568	20879	18932	21405	20503
11	18869	21716	18995	21094	20668	21163	20789	21036	21251	21249	21776	18932	21405	20503
12	18869	21716	18995	21094	21368	21163	20789	21036	19154	21249	21776	18932	21405	20503
13	18869	21529	18654	21094	21405	19545	20789	19418	19154	21249	21776	18932	21405	20503
14	18869	18301	18301	21094	21405	18780	21246	18806	18364	21249	20161	18780	21405	20503
15	18869	18301	18301	21094	19045	18301	19051	18806	18301	21218	18301	18301	19045	20503
16	18460	18301	18301	20596	18301	18301	18301	18938	18301	18301	18301	18301	18301	20503
17	18301	18301	18301	18301	18301	18301	18301	19564	18301	18301	18301	18301	18301	20503

Table (Ap4-7): Available metal recovery rate for each mixing scenario, (kIb/h), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	42.551	42.395
2	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	42.551	42.395
3	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	42.551	42.395
4	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	42.551	42.395
5	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	42.551	42.395
6	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	42.551	42.395
7	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	42.551	42.395
8	39.267	41.238	38.562	43.104	41.843	40.559	41.582	41.066	39.590	42.472	41.555	37.194	41.150	42.395
9	39.267	40.980	38.562	42.793	41.843	40.559	41.582	41.066	39.590	42.472	41.555	44.726	40.210	42.395
10	40.679	39.423	40.088	40.924	41.843	40.559	41.582	41.066	39.590	42.169	41.277	46.024	40.210	42.395
11	46.032	39.423	45.971	40.924	41.843	40.559	41.582	41.066	40.252	40.580	40.364	46.024	40.210	42.395
12	46.032	39.423	45.971	40.924	40.301	40.559	41.582	41.066	46.545	40.580	40.364	46.024	40.210	42.395
13	46.032	39.906	45.820	40.924	40.210	44.682	41.582	44.805	46.545	40.580	40.364	46.024	40.210	42.395
14	46.032	44.832	44.832	40.924	40.210	45.989	40.589	46.008	45.043	40.580	42.658	45.989	40.210	42.395
15	46.032	44.832	44.832	40.924	44.098	44.832	44.091	46.008	44.832	40.653	44.832	44.832	44.098	42.395
16	45.345	44.832	44.832	41.921	44.832	44.832	44.832	46.018	44.832	44.832	44.832	44.832	44.832	42.395
17	44.832	44.832	44.832	44.832	44.832	44.832	44.832	43.805	44.832	44.832	44.832	44.832	44.832	42.395

Table (Ap4-8): Specific CO_2 emission (Plant) for each mixing scenario, ($kg/t_{(processed)}$), (*Avg-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.408	1.422
2	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.408	1.422
3	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.408	1.422
4	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.408	1.422
5	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.408	1.422
6	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.408	1.422
7	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.408	1.422
8	1.504	1.443	1.513	1.399	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.548	1.457	1.422
9	1.504	1.453	1.513	1.410	1.434	1.469	1.443	1.460	1.491	1.417	1.439	1.357	1.486	1.422
10	1.468	1.508	1.477	1.464	1.434	1.469	1.443	1.460	1.491	1.427	1.449	1.312	1.486	1.422
11	1.307	1.508	1.316	1.464	1.434	1.469	1.443	1.460	1.475	1.475	1.512	1.312	1.486	1.422
12	1.307	1.508	1.316	1.464	1.483	1.469	1.443	1.460	1.327	1.475	1.512	1.312	1.486	1.422
13	1.307	1.495	1.292	1.464	1.486	1.355	1.443	1.346	1.327	1.475	1.512	1.312	1.486	1.422
14	1.307	1.267	1.267	1.464	1.486	1.301	1.475	1.303	1.272	1.475	1.398	1.301	1.486	1.422
15	1.307	1.267	1.267	1.464	1.319	1.267	1.320	1.303	1.267	1.473	1.267	1.267	1.319	1.422
16	1.278	1.267	1.267	1.429	1.267	1.267	1.267	1.312	1.267	1.267	1.267	1.267	1.267	1.422
17	1.267	1.267	1.267	1.267	1.267	1.267	1.267	1.356	1.267	1.267	1.267	1.267	1.267	1.422

Appendix 5: Mass flow and power requirements for each mixing scenario, (*Avg-Method*)

Table (Ap5-1): Annual mass flow and power requirements for (mixing scenario 2), (*Avg-2*).

Item /Y	Feeding rate (P. C.), t/h	Feeding rate (C. G.), t/h	Feeding rate (F. G.), t/h	Primary crushers bypass, t/h	Coarse grinding bypass, t/h	Fine grinding bypass, t/h	Power required (P. C.), kW	Power required (C. G.), kW	Power required (F. G.), kW
1	1920	2037	2181	342	226	81	730	9275	20795
2	1920	2037	2181	342	226	81	730	9275	20795
3	1920	2037	2181	342	226	81	730	9275	20795
4	1920	2037	2181	342	226	81	730	9275	20795
5	1920	2037	2181	342	226	81	730	9275	20795
6	1920	2037	2181	342	226	81	730	9275	20795
7	1920	2037	2181	342	226	81	730	9275	20795
8	1920	2037	2181	342	226	81	730	9275	20795
9	1921	2037	2181	342	225	81	733	9345	20942
10	1923	2038	2181	340	225	81	754	9715	21716
11	1923	2038	2181	340	225	81	754	9715	21716
12	1923	2038	2181	340	225	81	754	9715	21716
13	1921	2037	2181	341	225	81	749	9625	21529
14	1901	2028	2175	362	234	87	662	8082	18301
15	1901	2028	2175	362	234	87	662	8082	18301
16	1901	2028	2175	362	234	87	662	8082	18301
17	1901	2028	2175	362	234	87	662	8082	18301

Table (Ap5-2): Annual mass flow and power requirements for (mixing scenario 3), (*Avg-3*).

Item /Y	Feeding rate (P. C.), t/h	Feeding rate (C. G.), t/h	Feeding rate (F. G.), t/h	Primary crushers bypass, t/h	Coarse grinding bypass, t/h	Fine grinding bypass, t/h	Power required (P. C.), kW	Power required (C. G.), kW	Power required (F. G.), kW
1	1925	2039	2182	337	223	80	756	9749	21786
2	1925	2039	2182	337	223	80	756	9749	21786
3	1925	2039	2182	337	223	80	756	9749	21786
4	1925	2039	2182	337	223	80	756	9749	21786
5	1925	2039	2182	337	223	80	756	9749	21786
6	1925	2039	2182	337	223	80	756	9749	21786
7	1925	2039	2182	337	223	80	756	9749	21786
8	1925	2039	2182	337	223	80	756	9749	21786
9	1925	2039	2182	337	223	80	756	9749	21786
10	1922	2038	2182	340	225	81	743	9508	21283
11	1909	2032	2178	353	231	84	681	8414	18995
12	1909	2032	2178	353	231	84	681	8414	18995
13	1905	2030	2177	357	232	86	672	8251	18654
14	1901	2028	2175	362	234	87	662	8082	18301
15	1901	2028	2175	362	234	87	662	8082	18301
16	1901	2028	2175	362	234	87	662	8082	18301
17	1901	2028	2175	362	234	87	662	8082	18301

Appendix 6: Metal recovery and financial data for each mixing scenario, (Avg-*Method*)

Table (Ap6-1): Annual metal recovery and judgment financial data for (mixing scenario 2), (*Avg-2*).

Item /Y	Available metal recovery rate, klb/h	30% Concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	41.238	62	225.356	28.274	113.794	98.951	-104.249
2	41.238	62	225.356	28.274	113.794	86.045	-18.204
3	41.238	62	225.356	28.274	113.794	74.822	56.618
4	41.238	62	225.356	28.274	113.794	65.062	121.680
5	41.238	62	225.356	28.274	113.794	56.576	178.256
6	41.238	62	225.356	28.274	113.794	49.196	227.452
7	41.238	62	225.356	28.274	113.794	42.779	270.231
8	41.238	62	225.356	28.274	113.794	37.200	307.431
9	40.980	62	221.025	28.477	109.239	31.053	338.483
10	39.423	60	197.877	29.545	84.914	20.989	359.473
11	39.423	60	197.877	29.545	84.914	18.252	377.724
12	39.423	60	197.877	29.545	84.914	15.871	393.595
13	39.906	60	201.013	29.287	88.413	14.370	407.965
14	44.832	68	239.671	24.827	133.022	18.800	426.765
15	44.832	68	239.671	24.827	133.022	16.348	443.112
16	44.832	68	239.671	24.827	133.022	14.215	457.328
17	44.832	68	88.750	9.194	49.258	4.577	461.905

Table (Ap6-2): Annual metal recovery and judgment financial data for (mixing scenario 3), (*Avg-3*).

Item /Y	Available metal recovery rate, klb/h	30% Concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	38.562	58	201.569	29.643	88.225	76.718	-126.483
2	38.562	58	201.569	29.643	88.225	66.711	-59.772
3	38.562	58	201.569	29.643	88.225	58.009	-1.762
4	38.562	58	201.569	29.643	88.225	50.443	48.681
5	38.562	58	201.569	29.643	88.225	43.864	92.544
6	38.562	58	201.569	29.643	88.225	38.142	130.686
7	38.562	58	201.569	29.643	88.225	33.167	163.853
8	38.562	58	201.569	29.643	88.225	28.841	192.694
9	38.562	58	201.569	29.643	88.225	25.079	217.773
10	40.088	61	212.764	28.948	100.375	24.811	242.585
11	45.971	70	260.777	25.787	152.559	32.792	275.376
12	45.971	70	260.777	25.787	152.559	28.514	303.890
13	45.820	69	252.574	25.316	145.137	23.589	327.479
14	44.832	68	239.671	24.827	133.022	18.800	346.279
15	44.832	68	239.671	24.827	133.022	16.348	362.627
16	44.832	68	239.671	24.827	133.022	14.215	376.842
17	44.832	68	88.750	9.194	49.258	4.577	381.419

Appendix 7: Mass flow, economic and environmental output for the (*Crit-Method*) overall scenarios

Table (Ap7-1): Annual income for each mixing scenario, (M$), (*Crit-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	250.734	228.319
2	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	250.734	228.319
3	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	250.734	228.319
4	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	250.734	228.319
5	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	250.734	228.319
6	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	250.734	228.319
7	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	250.734	228.319
8	214.781	240.178	219.747	245.427	234.985	224.769	229.803	219.747	222.251	240.178	237.574	209.872	221.248	228.319
9	214.781	233.953	219.747	239.135	234.985	224.769	229.803	219.747	222.251	240.178	237.574	247.702	205.020	228.319
10	221.498	202.615	226.541	207.439	234.985	224.769	229.803	219.747	222.251	233.953	231.384	256.097	205.020	228.319
11	253.408	202.615	258.800	207.439	234.985	224.769	229.803	219.747	225.275	206.227	207.924	256.097	205.020	228.319
12	253.408	202.615	258.800	207.439	206.469	224.769	229.803	219.747	262.608	206.227	207.924	256.097	205.020	228.319
13	253.408	204.056	244.373	207.439	205.020	247.278	229.803	242.009	262.608	206.227	207.924	256.097	205.020	228.319
14	253.408	229.849	229.849	207.439	205.020	249.668	211.290	250.734	232.410	206.227	214.781	249.668	205.020	228.319
15	253.408	229.849	229.849	207.439	223.760	229.849	223.710	250.734	229.849	206.469	229.849	229.849	223.760	228.319
16	236.330	229.849	229.849	211.339	229.849	229.849	229.849	256.367	229.849	229.849	229.849	229.849	229.849	228.319
17	85.113	85.113	85.113	85.113	85.113	85.113	85.113	105.078	85.113	85.113	85.113	85.113	85.113	84.5466
Total	**3743.010**	**3741.938**	**3740.897**	**3743.637**	**3740.066**	**3738.985**	**3737.401**	**3741.886**	**3750.222**	**3741.894**	**3752.914**	**3745.545**	**3745.228**	**3737.651**

Table (Ap7-2): Total energy annual cost for each mixing scenario, (M$), (*Crit-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.066	46.0593
2	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.066	46.0593
3	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.066	46.0593
4	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.066	46.0593
5	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.066	46.0593
6	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.066	46.0593
7	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.066	46.0593
8	46.101	46.080	46.110	46.057	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.125	46.076	46.0593
9	46.101	46.083	46.110	46.059	46.071	46.086	46.073	46.077	46.098	46.064	46.076	46.031	46.082	46.0593
10	46.083	46.094	46.092	46.070	46.071	46.086	46.073	46.077	46.098	46.066	46.078	46.009	46.082	46.0593
11	46.005	46.094	46.014	46.070	46.071	46.086	46.073	46.077	46.090	46.076	46.091	46.009	46.082	46.0593
12	46.005	46.094	46.014	46.070	46.082	46.086	46.073	46.077	46.013	46.076	46.091	46.009	46.082	46.0593
13	46.005	46.087	45.989	46.070	46.082	46.030	46.073	46.021	46.013	46.076	46.091	46.009	46.082	46.0593
14	46.005	45.964	45.964	46.070	46.082	45.998	46.080	46.000	45.968	46.076	46.035	45.998	46.082	46.0593
15	46.005	45.964	45.964	46.070	45.992	45.964	45.992	46.000	45.964	46.075	45.964	45.964	45.992	46.0593
16	45.975	45.964	45.964	46.051	45.964	45.964	45.964	46.010	45.964	45.964	45.964	45.964	45.964	46.0593
17	17.020	17.020	17.020	17.020	17.020	17.020	17.020	17.054	17.020	17.020	17.020	17.020	17.020	17.0557
Total	**754.007**	**754.007**	**754.008**	**754.004**	**754.005**	**754.007**	**754.005**	**754.005**	**754.010**	**754.005**	**754.015**	**754.009**	**754.005**	**754.0045**

Appendix 8: Mass flow and power requirements for each mixing scenario, (*Crit-Method*)

Table (Ap8-1): Annual mass flow and power requirements for (mixing scenario 1), (*Crit-1*).

Item /Y	Feeding rate (P. C.), t/h	Feeding rate (C. G.), t/h	Feeding rate (F. G.), t/h	Primary crushers bypass, t/h	Coarse grinding bypass, t/h	Fine grinding bypass, t/h	Power required (P. C.), kW	Power required (C. G.), kW	Power required (F. G.), kW
1	1924	2038	2183	339	224	80	825	11022	38371
2	1924	2038	2183	339	224	80	825	11022	38371
3	1924	2038	2183	339	224	80	825	11022	38371
4	1924	2038	2183	339	224	80	825	11022	38371
5	1924	2038	2183	339	224	80	825	11022	38371
6	1924	2038	2183	339	224	80	825	11022	38371
7	1924	2038	2183	339	224	80	825	11022	38371
8	1924	2038	2183	339	224	80	825	11022	38371
9	1924	2038	2183	339	224	80	825	11022	38371
10	1921	2037	2182	342	225	80	825	11015	38360
11	1908	2031	2179	355	231	83	821	10984	38309
12	1908	2031	2179	355	231	83	821	10984	38309
13	1908	2031	2179	355	231	83	821	10984	38309
14	1908	2031	2179	355	231	83	821	10984	38309
15	1908	2031	2179	355	231	83	821	10984	38309
16	1903	2029	2178	360	233	84	819	10972	38290
17	1901	2028	2178	362	234	85	819	10968	38283

Appendix 9: Metal recovery and financial data for each mixing scenario, (Crit-*Method*)

Table (Ap9-1): Annual metal recovery and judgment financial data for (mixing scenario 1), (*Crit-1*).

Item /Y	Available metal recovery rate, klb/h	30% Concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	41.561	63	214.781	46.101	85.137	74.032	-129.168
2	41.561	63	214.781	46.101	85.137	64.376	-64.792
3	41.561	63	214.781	46.101	85.137	55.979	-8.812
4	41.561	63	214.781	46.101	85.137	48.678	39.865
5	41.561	63	214.781	46.101	85.137	42.328	82.193
6	41.561	63	214.781	46.101	85.137	36.807	119.001
7	41.561	63	214.781	46.101	85.137	32.006	151.007
8	41.561	63	214.781	46.101	85.137	27.832	178.838
9	41.561	63	214.781	46.101	85.137	24.201	203.040
10	42.206	64	221.498	46.083	92.121	22.771	225.811
11	45.144	68	253.408	46.005	125.090	26.887	252.698
12	45.144	68	253.408	46.005	125.090	23.380	276.078
13	45.144	68	253.408	46.005	125.090	20.331	296.409
14	45.144	68	253.408	46.005	125.090	17.679	314.088
15	45.144	68	253.408	46.005	125.090	15.373	329.461
16	43.597	66	236.330	45.975	108.400	11.584	341.045
17	42.995	65	85.113	17.020	37.794	3.512	344.557

Table (Ap9-2): Annual metal recovery and judgment financial data for (mixing scenario 2), (*Crit-2*).

Item /Y	Available metal recovery rate, klb/h	30% Concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	43.950	66	240.178	46.080	110.810	96.356	-106.844
2	43.950	66	240.178	46.080	110.810	83.788	-23.056
3	43.950	66	240.178	46.080	110.810	72.859	49.803
4	43.950	66	240.178	46.080	110.810	63.356	113.159
5	43.950	66	240.178	46.080	110.810	55.092	168.251
6	43.950	66	240.178	46.080	110.810	47.906	216.157
7	43.950	66	240.178	46.080	110.810	41.657	257.814
8	43.950	66	240.178	46.080	110.810	36.224	294.038
9	43.377	66	233.953	46.083	104.562	29.723	323.761
10	40.367	61	202.615	46.094	73.102	18.070	341.831
11	40.367	61	202.615	46.094	73.102	15.713	357.544
12	40.367	61	202.615	46.094	73.102	13.663	371.207
13	40.510	61	204.056	46.087	74.655	12.134	383.341
14	42.995	65	229.849	45.964	102.064	14.425	397.765
15	42.995	65	229.849	45.964	102.064	12.543	410.308
16	42.995	65	229.849	45.964	102.064	10.907	421.215
17	42.995	65	85.113	17.020	37.794	3.512	424.727

Appendix 10: Mass flow, economic and environmental output for the (*Org-Method*) overall scenarios

Table (Ap10-1): Fine grinding by-pass for each mixing scenario, (t/h), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	81	81	80	82	82	81	82	82	81	82	82	79	82	82
2	81	81	80	82	82	81	82	82	81	82	82	79	82	82
3	81	81	80	82	82	81	82	82	81	82	82	79	82	82
4	81	81	80	82	82	81	82	82	81	82	82	79	82	82
5	81	81	80	82	82	81	82	82	81	82	82	79	82	82
6	81	81	80	82	82	81	82	82	81	82	82	79	82	82
7	81	81	80	82	82	81	82	82	81	82	82	79	82	82
8	81	81	80	82	82	81	82	82	81	82	82	79	82	82
9	81	81	80	82	82	81	82	82	81	82	82	84	82	82
10	81	81	81	82	82	81	82	82	81	82	82	85	82	82
11	85	81	84	82	82	81	82	82	81	82	82	85	82	82
12	85	81	84	82	82	81	82	82	85	82	82	85	82	82
13	85	81	86	82	82	84	82	84	85	82	82	85	82	82
14	85	87	87	82	82	85	82	85	87	82	84	85	82	82
15	85	87	87	82	86	87	86	85	87	82	87	87	86	82
16	86	87	87	83	87	87	87	85	87	87	87	87	87	82
17	87	87	87	87	87	87	87	82	87	87	87	87	87	82

Table (Ap10-2): Energy rate consumption (primary crushing) for each mixing scenario, (kW), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	498	475	501	458	471	484	474	481	493	464	473	515	461	723
2	498	475	501	458	471	484	474	481	493	464	473	515	461	723
3	498	475	501	458	471	484	474	481	493	464	473	515	461	723
4	498	475	501	458	471	484	474	481	493	464	473	515	461	723
5	498	475	501	458	471	484	474	481	493	464	473	515	461	723
6	498	475	501	458	471	484	474	481	493	464	473	515	461	723
7	498	475	501	458	471	484	474	481	493	464	473	515	461	723
8	498	475	501	458	471	484	474	481	493	464	473	515	480	723
9	498	478	501	462	471	484	474	481	493	464	473	441	491	723
10	484	499	488	482	471	484	474	481	493	468	477	424	491	723
11	422	499	426	482	471	484	474	481	487	487	500	424	491	723
12	422	499	426	482	490	484	474	481	429	487	500	424	491	723
13	422	494	416	482	491	441	474	437	429	487	500	424	491	723
14	422	407	407	482	491	420	487	421	408	487	457	420	491	723
15	422	407	407	482	427	407	427	421	407	486	407	407	427	723
16	411	407	407	469	407	407	407	424	407	407	407	407	407	723
17	407	407	407	407	407	407	407	441	407	407	407	407	407	723

Table (Ap10-3): Energy rate consumption (coarse grinding) for each mixing scenario, (kW), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9041	9140
2	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9041	9140
3	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9041	9140
4	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9041	9140
5	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9041	9140
6	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9041	9140
7	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9041	9140
8	9693	9279	9754	8981	9218	9456	9276	9395	9605	9099	9249	9992	9378	9140
9	9693	9350	9754	9052	9218	9456	9276	9395	9605	9099	9249	8690	9572	9140
10	9452	9721	9513	9423	9218	9456	9276	9395	9605	9170	9319	8384	9572	9140
11	8354	9721	8415	9423	9218	9456	9276	9395	9498	9498	9743	8384	9572	9140
12	8354	9721	8415	9423	9555	9456	9276	9395	8483	9498	9743	8384	9572	9140
13	8354	9632	8251	9423	9572	8679	9276	8618	8483	9498	9743	8384	9572	9140
14	8354	8082	8082	9423	9572	8312	9496	8324	8112	9498	8976	8312	9572	9140
15	8354	8082	8082	9423	8439	8082	8442	8324	8082	9483	8082	8082	8439	9140
16	8158	8082	8082	9185	8082	8082	8082	8387	8082	8082	8082	8082	8082	9140
17	8082	8082	8082	8082	8082	8082	8082	8687	8082	8082	8082	8082	8082	9140

Table (Ap10-4): Energy rate consumption (fine grinding) for each mixing scenario, (kW), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	24733	23850
2	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	24733	23850
3	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	24733	23850
4	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	24733	23850
5	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	24733	23850
6	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	24733	23850
7	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	24733	23850
8	28090	26192	29108	23715	25174	26632	25318	25614	27870	24444	25683	30567	25570	23850
9	28090	26368	29108	23891	25174	26632	25318	25614	27870	24444	25683	20965	26054	23850
10	26320	27293	27338	24816	25174	26632	25318	25614	27870	24621	25859	18763	26054	23850
11	18254	27293	19272	24816	25174	26632	25318	25614	27083	25435	26934	18763	26054	23850
12	18254	27293	19272	24816	26010	26632	25318	25614	19205	25435	26934	18763	26054	23850
13	18254	26550	16523	24816	26054	20927	25318	19909	19205	25435	26934	18763	26054	23850
14	18254	13673	13673	24816	26054	17541	25865	17745	14182	25435	21102	17541	26054	238500
15	18254	13673	13673	24816	16644	13673	16669	17745	13673	25311	13673	13673	16644	2385
16	14955	13673	13673	22835	13673	13673	13673	18813	13673	13673	13673	13673	13673	23850
17	13673	13673	13673	13673	13673	13673	13673	23853	13673	13673	13673	13673	13673	23850

Table (Ap10-5): Available metal recovery rate for each mixing scenario, (kIb/h), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	44.643	42.580
2	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	44.643	42.580
3	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	44.643	42.580
4	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	44.643	42.580
5	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	44.643	42.580
6	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	44.643	42.580
7	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	44.643	42.580
8	41.369	43.716	41.863	44.149	43.222	42.296	42.744	41.802	42.079	43.685	43.469	40.936	41.944	42.580
9	41.369	43.148	41.863	43.581	43.222	42.296	42.744	41.802	42.079	43.685	43.469	44.305	40.381	42.580
10	41.981	40.165	42.475	40.597	43.222	42.296	42.744	41.802	42.079	43.117	42.901	45.014	40.381	42.580
11	44.767	40.165	45.261	40.597	43.222	42.296	42.744	41.802	42.351	40.489	40.677	45.014	40.381	42.580
12	44.767	40.165	45.261	40.597	40.523	42.296	42.744	41.802	45.588	40.489	40.677	45.014	40.381	42.580
13	44.767	40.295	43.928	40.597	40.381	44.267	42.744	43.773	45.588	40.489	40.677	45.014	40.381	42.580
14	44.767	42.545	42.545	40.597	40.381	44.422	40.992	44.520	42.792	40.489	41.247	44.422	40.381	42.580
15	44.767	42.545	42.545	40.597	42.025	42.545	42.021	44.520	42.545	40.511	42.545	42.545	42.025	42.580
16	43.167	42.545	42.545	40.944	42.545	42.545	42.545	45.039	42.545	42.545	42.545	42.545	42.545	42.580
17	42.545	42.545	42.545	42.545	42.545	42.545	42.545	47.484	42.545	42.545	42.545	42.545	42.545	42.580

Table (Ap10-6): Specific CO_2 emission (Plant) for each mixing scenario, ($kg/t_{(processed)}$), (*Org-Method*)

Sc./Y	1-302050	2-205030	3-203050	4-305020	5-304030	6-303040	7-333333	8-402040	9-253045	10-304525	11-254530	12-202060	13-206020	14-432730
1	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.454	1.343
2	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.454	1.343
3	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.454	1.343
4	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.454	1.343
5	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.454	1.343
6	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.454	1.343
7	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.454	1.343
8	1.625	1.526	1.671	1.409	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.743	1.505	1.343
9	1.625	1.537	1.671	1.419	1.481	1.553	1.490	1.507	1.612	1.445	1.504	1.279	1.534	1.343
10	1.540	1.593	1.585	1.475	1.481	1.553	1.490	1.507	1.612	1.455	1.514	1.173	1.534	1.343
11	1.151	1.593	1.196	1.475	1.481	1.553	1.490	1.507	1.574	1.504	1.563	1.173	1.534	1.343
12	1.151	1.593	1.196	1.475	1.531	1.553	1.490	1.507	1.185	1.504	1.563	1.173	1.534	1.343
13	1.151	1.557	1.073	1.475	1.534	1.278	1.490	1.232	1.185	1.504	1.563	1.173	1.534	1.343
14	1.151	0.945	0.945	1.475	1.534	1.119	1.522	1.128	0.968	1.504	1.298	1.119	1.534	1.343
15	1.151	0.945	0.945	1.475	1.086	0.945	1.088	1.128	0.945	1.498	0.945	0.945	1.086	1.343
16	1.003	0.945	0.945	1.381	0.945	0.945	0.945	1.176	0.945	0.945	0.945	0.945	0.945	1.343
17	0.945	0.945	0.945	0.945	0.945	0.945	0.945	1.401	0.945	0.945	0.945	0.945	0.945	1.343

Appendix 11: Deatailed power requirements for each mixing scenario, (*Org-Method*)

Table (Ap11-1): Annual detailed power requirements (kW) for (mixing scenario 2), (*Org-2*).

Item /Y	Power required (PC)-1	Power required (PC)-2	Power required (PC)-3	Total power (PC)	Power required (CG)-1	Power required (CG)-2	Power required (CG)-3	Total power (CG)	Power required (FG)-1	Power required (FG)-2	Power required (FG)-3	Total power (FG)
1	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
2	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
3	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
4	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
5	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
6	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
7	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
8	81	221	172	475	1616	4344	3319	9279	2735	11926	11531	26192
9	98	185	195	478	1940	3649	3761	9350	3281	10018	13068	26368
10	183	0	316	499	3637	0	6085	9721	6153	0	21140	27293
11	183	0	316	499	3637	0	6085	9721	6153	0	21140	27293
12	183	0	316	499	3637	0	6085	9721	6153	0	21140	27293
13	195	0	299	494	3879	0	5753	9632	6563	0	19987	26550
14	407	0	0	407	8082	0	0	8082	13673	0	0	13673
15	407	0	0	407	8082	0	0	8082	13673	0	0	13673
16	407	0	0	407	8082	0	0	8082	13673	0	0	13673
17	407	0	0	407	8082	0	0	8082	13673	0	0	13673

Table (Ap11-2): Annual detailed power requirements (kW) for (mixing scenario 3), (*Org-3*).

Item /Y	Power required (PC)-1	Power required (PC)-2	Power required (PC)-3	Total power (PC)	Power required (CG)-1	Power required (CG)-2	Power required (CG)-3	Total power (CG)	Power required (FG)-1	Power required (FG)-2	Power required (FG)-3	Total power (FG)
1	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
2	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
3	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
4	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
5	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
6	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
7	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
8	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
9	81	132	287	501	1616	2606	5531	9754	2735	7156	19218	29108
10	100	152	236	488	1980	2997	4536	9513	3350	8229	15759	27338
11	183	243	0	426	3637	4778	0	8415	6153	13119	0	19272
12	183	243	0	426	3637	4778	0	8415	6153	13119	0	19272
13	293	124	0	416	5819	2432	0	8251	9844	6679	0	16523
14	407	0	0	407	8082	0	0	8082	13673	0	0	13673
15	407	0	0	407	8082	0	0	8082	13673	0	0	13673
16	407	0	0	407	8082	0	0	8082	13673	0	0	13673
17	407	0	0	407	8082	0	0	8082	13673	0	0	13673

Appendix 12: Metal recovery and financial data for each mixing scenario, (Org-*Method*)

Table (Ap12-1): Annual metal recovery and judgment financial data for (mixing scenario 2), (*Org-2*).

Item /Y	Metal recovery rate (1), klb/h	Metal recovery rate (2), klb/h	Metal recovery rate (3), klb/h	Available metal recovery rate, klb/h	Total 30% concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	108.242	-94.958
2	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	94.123	-0.835
3	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	81.847	81.012
4	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	71.171	152.183
5	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	61.888	214.071
6	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	53.815	267.886
7	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	46.796	314.682
8	8.509	23.742	11.465	43.716	66	240.998	33.232	124.478	40.692	355.374
9	10.211	19.943	12.994	43.148	65	234.796	33.462	118.025	33.550	388.924
10	19.145	0.000	21.020	40.165	61	202.235	34.672	84.145	20.799	409.724
11	19.145	0.000	21.020	40.165	61	202.235	34.672	84.145	18.086	427.810
12	19.145	0.000	21.020	40.165	61	202.235	34.672	84.145	15.727	443.537
13	20.421	0.000	19.873	40.294	61	203.610	33.903	86.393	14.041	457.579
14	42.545	0.000	0.000	42.545	64	227.444	20.578	125.044	17.672	475.251
15	42.545	0.000	0.000	42.545	64	227.444	20.578	125.044	15.367	490.618
16	42.545	0.000	0.000	42.545	64	227.444	20.578	125.044	13.363	503.981
17	42.545	0.000	0.000	42.545	64	84.222	7.620	46.304	4.303	508.284

Table (Ap12-2): Annual metal recovery and judgment financial data for (mixing scenario 3), (*Org-3*).

Item /Y	Metal recovery rate (1), klb/h	Metal recovery rate (2), klb/h	Metal recovery rate (3), klb/h	Available metal recovery rate, klb/h	Total 30% concent. ore rate, t/h	Income, M$/Y	Energy cost, M$/Y	Total cash flow, M$/Y	Present value, M$	Cum. present value, M$
1	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	87.685	-115.515
2	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	76.248	-39.267
3	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	66.303	27.036
4	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	57.655	84.691
5	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	50.134	134.825
6	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	43.595	178.420
7	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	37.909	216.329
8	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	32.964	249.293
9	8.509	14.245	19.109	41.863	63	220.909	36.370	100.838	28.665	277.958
10	10.423	16.382	15.669	42.474	64	227.492	34.511	109.539	27.077	305.034
11	19.145	26.116	0.000	45.261	68	257.480	26.042	149.007	32.028	337.062
12	19.145	26.116	0.000	45.261	68	257.480	26.042	149.007	27.851	364.913
13	30.632	13.296	0.000	43.928	66	242.734	23.360	137.253	22.308	387.220
14	42.545	0.000	0.000	42.545	64	227.444	20.578	125.044	17.672	404.893
15	42.545	0.000	0.000	42.545	64	227.444	20.578	125.044	15.367	420.260
16	42.545	0.000	0.000	42.545	64	227.444	20.578	125.044	13.363	433.623
17	42.545	0.000	0.000	42.545	64	84.222	7.620	46.304	4.303	437.926

Declaration

I hereby declare that I completed this work without any improper help from a third party and without using any aids other than those cited. All ideas derived directly or indirectly from other sources are identified as such.

Persons other than those above did not contribute to the writing of this thesis. I did not seek the help of a professional doctorate-consultant. Only those persons identified as having done so received any financial payment from me for any work done for me.

This thesis has not previously been published in the same or a similar form in Germany or abroad.

_____________________________ Date, signature